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# MINING OF MINERAL DEPOSITS

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# MINING OF MINERAL DEPOSITS

A TEXTBOOK

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# OPENING UP OF MINERAL DEPOSITS

CHAPTER I

# BASIC CONCEPTS AND TERMINOLOGY

## 1. Useful Minerals and Their Occurrence

Minerals and rocks extracted from the earth's crust for purposes of utilisation are called *useful* or *valuable.*

*Extraction* is the aggregate of industrial operations required to remove useful minerals from the earth's crust.

*Mine output* is the amount of useful mineral recovered from the earth's crust within a certain period of time.

Accumulation of useful minerals in the earth's crust is termed *occurrence* or *deposit.*

Rocks with mineral occurrences are called *country rocks,* while those interlaying with valuable minerals are *gangues.*

The notion of useful mineral is a *relative one.*

In certain conditions a given mineral or rock may be extracted as a valuable mineral, while in others it is considered a gangue or waste. For example, immense quantities of limestone are mined in special quarries and employed as flux for pig-iron smelting in blast furnaces, whereas at coal pits and other mines workings sometimes intersect beds of limestone, which, however, is dumped as useless gangue, since, in this instance, in view of the small amount of rock recovered, its use would be economically unprofitable.

Minerals may be solid, liquid or gaseous.

The solid ones include mineral coals, metal ores, mineral salts, building materials and many other minerals; liquid ones—crude oil, brines and mineral waters. As an example of gaseous minerals we may cite natural gases, recovered through boreholes and subsequently burned or processed on the spot, or transported by pipeline over long distances (for instance, "Saratov gas", recovered in the middle reaches of the Volga, is transported to Moscow via gas mains over a distance of about 800 km).

With the passage of time and progress in science and technology the number of valuable minerals constantly increases. The discovery of radioactivity at the close of the 19th century has led to the inclusion of the ores of radioactive elements in the list of minerals. The same holds true for the ores of rare earths and new gaseous minerals, such as helium, recovered through boreholes together with other gases. There are many other examples we could cite.

Achievements in research sometimes lead to the reappraisal of the importance of minerals. Thus, a very important role in modern technology is played by aluminium which is obtained chiefly from bauxites. Before the development of the aluminium industry bauxites were sometimes considered poor iron ores.

By their industrial importance, and following in the main the pattern laid down by A. Fersman, minerals may now be classified as follows:

A. **Metal ores and metals proper** (iron, manganese, vanadium, chromium, gold, platinum, silver, lead, zinc, aluminium, tin, copper, nickel, tungsten, molybdenum, cobalt, titanium, beryllium, niobium, antimony, bismuth, mercury, etc.).

B. **Fuels** (coal, combustible shales, crude oil, natural gases, etc.).

It is noteworthy that in modern industry the minerals of this group are used to an ever-increasing extent not only as fuel but as raw materials for the production of a huge variety of chemicals.

C. **Nonmetallic minerals:**

a) *salts* (table salt, potassium and magnesium salts, saltpeter, natural sodium carbonate, sodium sulphate, etc.);

b) *abrasives*—grinding and honing materials for the processing of surfaces: emery, corundum, pumice, honing and polishing stones, flint; others that may be included in this group are diamonds and garnets, listed below as precious stones;

c) *ceramics*, *glass* and *insulation materials* (asbestos, dolomite, acid-resistant and refractory clays, quartz and quartzite, feldspar, mica, talc and many others);

d) *building materials* (asphalt, gypsum, anhydrite, slate, limestone, clays, sand, gravel, marble, stone building materials, various cement materials, etc.);

e) *miscellaneous industrial materials* (barite, graphite, pyrites, mineral paints, lithographic stone, magnesite, chalk, mineral wax, sulphur, tripoli, diatomite, etc.);

f) *mineral fertilisers* (apatite, phosphorite, potassium salts, etc.);

g) *precious, coloured and decorative or ornamental stones* (diamonds, aquamarine, tourmaline, garnet, opal, turquoise, varieties of quartz, amber, malachite, jaspers, etc.);

h) *technical stones* (Iceland spar, agate, rock crystal, piezoquartz, etc.);

i) *natural gases* (oxygen, nitrogen, argon and other rare gases, helium, methane, etc.);

j) *radioactive and rare elements and their compounds*—radium, uranium, lithium, rubidium, etc.

The above classification is, to a certain measure, conventional, for some minerals and rocks may be listed twice: for example, diamonds may be called abrasives and gems; potassium chloride compounds may be included both in the salt and mineral fertiliser groups, and so on.

Deposits of useful minerals are one of the forms of occurrence in the earth's crust of minerals and rocks and are, therefore, primarily a subject of geological study, or, to be more precise, of study by a special department of this extensive branch of science—*economic geology or teaching on mineral deposits*. Being one of the series of natural science disciplines, its particular aim is to create a natural classification of the subject-matter it is called upon to study. As in the study of other natural objects, this classification is of a *genetic* type, that is, based on features specific to the geological origin of mineral deposits.

However, for *mining purposes*, that is, for working mineral deposits, their *form* is of prime importance, regardless of the peculiarities incidental to their geological origin.

Below is a brief description of the forms of mineral deposits.

## 2. Assured Economic Value of Mineral Deposits

Mineral occurrences *suitable for mining* are said to have *an assured economic value*, that is, they are commercial or paying as distinct from those where this assurance is lacking and which are called noncommercial or nonpaying.

The concept of assured or commercial value is a relative one. The true economic value of a deposit depends not only upon its geological characteristics, but also on the importance its exploitation has for the national economy of the country as a whole.

There is a marked and essential difference between the definition of a deposit's assured or commercial value under socialism and that in capitalist conditions.

Under capitalism a mineral deposit is said to have an assured economic value when its exploitation brings profit.

In Soviet, socialist conditions a mineral deposit is considered paying or commercial when its exploitation is *warranted from the viewpoint of the benefits it brings to the entire national economy* and, first and foremost, with due account of the economic factor involved.

The factors determining the commercial value of a mineral deposit may be divided into two groups: 1) quantity and quality of reserves and the geological conditions attending the occurrence, and 2) possible importance of its exploitation for the national economy.

The factors of the first group are:

a) the presence of *workable* beds or deposits, economically suitable for mining because of their thickness, structure, physical and chemical features;

b) *sufficient reserves* of the valuable mineral to justify its mining;

c) *relatively regular nature of occurrence* (viewed geologically). A highly disturbed nature of the deposit and its division into small separate portions present considerable difficulties to mining. But even highly dislocated deposits may prove suitable for exploitation if they have sections of sufficient size to permit the adoption of a certain method of mining;

d) *the occurrence of a deposit in depth.* With the depth increasing, there may arise considerable difficulties in working a deposit. With depth the rock temperature rises at a certain rate (approximately by one degree per every 30-40 and sometimes more metres, from the mean annual temperature for the given area). In the case of gas-bearing deposits, primarily in coal beds, the increase of depth may cause a rapid expansion in the volume of liberated mine gases. Moreover, with depth the nature of gas evolution may become complicated; for example, instead of a regular, uniform outflow of gases there may be instances of violent sudden outbursts, accompanied by dislodgement of coal, sometimes in sizable proportions. The deeper the mine workings, the greater rock pressure may grow. Lastly, deep mining enhances difficulties in hoisting the mineral and gangue and complicates mine drainage and ventilation in underground workings.

For this reason it is customary to limit even the total potential (geological) reserves of extensive mine fields, where mineral deposits are known definitely to lie several kilometres below surface, to depths making it possible to overcome the above-mentioned difficulties with the aid of present-day techniques. Thus, in the Donets and Kuznetsk coal fields the potential (geological) reserves of coal are estimated to the depth of 1,500 metres below the sea level, that is, about 1,600-1,700 metres below ground surface, depending on local topography. There are mine fields where, because of the prevailing geological structure, minerals do not lie very deep, just a few scores of metres below surface (for example, the Cheremkhovo and Moscow coal fields). Ore deposits may be both deep (for instance, the iron ores of Krivoi Rog and the chalcopyrites in the Urals) and shallow (iron ore occurrences in the central regions of the European part of the U.S.S.R., manganese ores in the Nikopol area of the Ukraine, etc.).

In the U.S.S.R. the deepest workings now are in the Donets coal basin. The depth of the vertical shaft at the Shcheglovka-Deep Pit reaches 948 metres. The deepest mine workings abroad are in South Africa, India and Brazil;

e) unfavourable natural factors—*high water- and/or gas-bearing capacity of the deposit, unstable or insecure enclosing rocks*, etc. These factors complicate mining operations, but should not be regarded as a reason for abandoning the exploitation of the deposit. In certain instances, they tend to defer the ultimate industrial exploitation of a deposit.

The importance of the exploitation of any mineral deposit for the national economy is determined by the factors of the second group:

a) the extent to which the state needs a particular valuable mineral from the given deposit;

b) geographic location of the mine property;

c) availability of means of transport, or conditions favouring or complicating their construction;

d) available or possible sources of labour, food, power, materials and equipment.

All these factors differ diametrically under the capitalist and socialist economic systems.

Under capitalism the above-mentioned and other similar prerequisites for the exploitation of any deposit, as a rule, prevail irrespective of mining activities.

In Soviet conditions, if these prerequisites are absent at the initial stage of mining, they are created in a planned way. One example of such approach to the utilisation of the country's mineral wealth is the establishment by the Soviet Union of numerous major industrial districts: the Kuznetsk basin, Karaganda, Magnitogorsk (iron ores), Balkhash (copper ores) and many others.

After geological exploration and detailed prospecting had confirmed the presence of huge reserves of coal, particularly of coking one, in Karaganda, a railway line was built to link this remote region with the ultimate consumers of coal, well-appointed modern cities and workers' settlements rose, power stations were erected, adequate drinking, domestic and industrial water supplies were ensured, qualified labour force secured, etc.

All this was done in accordance with a previously elaborated state plan.

Even grander and more ambitious were the now implemented plans regarding the Kuznetsk basin and the Urals. These two huge industrial centres, lying far from each other, were conceived and developed in close coordination.

### 3. Importance of Exploration Work

Prior to the preparation of the mine layout and of the plan for its exploitation, the property must be *thoroughly explored.* The importance of exploration lies in the fact that, after a mineral deposit has been geologised and prospected, it is imperative to study its features in detail, both with regard to the quantity and quality of the valuable mineral and the conditions governing its working. This is necessary in order finally to establish the assured or commercial value of the property and to plan its layout and mining operations. The purpose of exploration work is to ascertain the type and shape of the mineral occurrence, the quantity and quality (grade) of mineral reserves, specific property of enclosing or country rocks and their water- and gas-bearing capacity. It should also furnish the data requisite for deciding on the concentration of the mineral.

It is not necessary and not always possible to explore the property in detail all at once. But this is absolutely indispensable, at least to the extent permitting substantiation of the estimates of mineral reserves in designing and building primary shafts. The better a mining property is explored, the easier it is to prepare a well-grounded *complex* (that is, coordinated) plan for its mining through shafts of the first and subsequent stages of operation.

Detailed exploration work is costly but unavoidable. If, to minimise expenditure and save time, shafts are sunk at insufficiently explored properties, the information on the quantity and quality of the mineral obtained through preliminary prospecting may find no confirmation and the capital outlays prove futile. Conversely, if exploration efforts are crowned with success, the sum spent on them would be insignificant compared with the value of the product extracted.

Hence, elaboration of plans for the operation of a mining enterprise requires a definite degree of exploration of its mineral reserves.

## 4. Mineral Reserves

Mineral resources belong to one of the so-called *categories of reserves* designated $A$, $B$ and $C$, according to the degree of exploration. A detailed definition of these concepts is given in textbooks on applied geology and prospecting. Briefly, it may be said that Category $A$ includes all *warranted* reserves, whose availability is proved by exploration and whose features have been studied in detail. Category $B$ covers reserves whose existence is considered *probable* after certain prospecting work and geological observation. Finally, there is Category $C$ which includes *possible* reserves, whose occurrence in the earth's crust may be presumed on the basis of geological considerations, substantiated in part by geophysical investigations and individual artificial or natural outcrops. Categories $A$ and $C$ are divided into subcategories $A_1$, $A_2$ and $C_1$ and $C_2$, according to the degree of exploration.

The estimated reserves of mineral deposits are subject to approval by the State Commission for Mineral Reserves of the Council of Ministers of the U.S.S.R. (S.C.R.). The approved reserves are called *inventory resources*. They include the reserves in categories $A$, $B$ and $C_1$, with those in the latter category taken account of separately from the resources in the superior categories. The inventory resources comprise reserves whose grade of mineral content meets industrial use specifications and whose quantity and mode of occurrence make their extraction profitable in the present-day conditions of technical and economic development.

In deciding the question of building new mines, and also in designing them, reserves belonging to categories $A$ and $B$ are alone considered as future prospects of "transferring" Category $C$ reserves to categories $A$ and $B$ in the course of further prospecting. For example, elaboration of preliminary project schemes for the construction of coal mines and open pits is permitted only on the basis of confirmed $A_2+B+C_1$ reserves, provided the available reserves of categories $A_2+B$ constitute no less than 50 per cent of total resources. For other minerals these proportions may be different.

## 5. Workable Reserves. Overall Losses of the Mineral During Mining Operations

In designing the mining enterprises one must base all estimates and calculations on *workable* mineral reserves. These involve amounts which can be readily recovered from the deposit. In other words, workable reserves are equal to the inventory ones minus *overall losses* as envisaged by the scheme of operations.

The reasons for the losses of the mineral and their percentage are closely linked with methods of mining and, therefore, will be dealt with in detail in subsequent chapters of this book. It may be said, however, that the chief causes are: 1) abandonment of the mineral in mined-out workings due to faulty or inadequate methods of operation in the form of fines, protective pillars, "safety" arch and floor pillars, etc.; 2) abandonment of the valuable mineral in pillars near mine workings; 3) abandonment of the mineral in "safety" pillars under surface structures and water basins whose drainage is either unfeasible or inexpedient.

In some instances the amount of loss ranges from a few to scores of per cent of the total inventory reserves.The magnitude of losses is determined by the *coefficient of losses*, that is, by the ratio between the amount of the mineral lost and its inventory reserves. If the figure involves the loss for the whole of the mine field it is called *coefficient of overall losses*. When it is a question of operational losses caused by a mining method, the losses are estimated by using the *coefficient of exploitation losses*. Hence, the overall losses at every mine are always somewhat in excess of the exploitation losses. The amount of mineral removed from the deposit is determined by the *coefficient of recovery*, which is indicative of the percentage of inventory reserves actually mined. Consequently, if we take inventory reserves as 1, the coefficient of recovery will equal 1 minus the coefficient of losses, and vice versa. For example, with losses amounting to 8 per cent, the coefficient of losses would come to 0.08, while that of recovery will equal 0.92. As in the case of losses, there are also a total *coefficient of recovery* for the entire mine and a *coefficient of exploitation recovery*.

The losses of the mineral may be not only quantitative, but *qualitative* too, the latter when the grade of the mineral is permitted to deteriorate in the process of mining. For instance, during the exploitation of ore bodies barren rocks enclosing useful mineral may become mixed with ore broken in stopes, thus *diluting* it, that is, lowering the content of its valuable components.

The losses of useful mineral in amounts exceeding permissible limits prejudice the interests of the national economy. Once incurred, these losses by their very nature are *irretrievable*. Instances of *repeated* working of deposits whose initial exploitation was attended by high losses and waste of mineral are rare exceptions to this rule. There have been cases of deposits previously mined by the underground method being later worked as open pits, this making it possible to recover the useful mineral formerly abandoned in pillars. There have also been instances of ore being recovered from pillars in old mined-out rooms where because of insufficient knowledge and experience in the past, protective pillars of unnecessarily large size had been abandoned. Such cases, however, are rare exceptions and, as a rule, losses which occurred during mining operations cannot possibly be made good again.

The adverse consequences of such losses are not limited only to squandering natural wealth.

Construction of new shafts requires considerable capital outlays, whose *depreciation* in the form of sinking fund constitutes a sizable part of the cost of mine production. The greater the loss, the lower is the percentage of recoverable reserves, the larger is capital expenditure per unit of output.

When the factual losses at an operating mine surpass the level envisaged in its operational plan, the result may be underfulfilment of production schedule.

If a mine builds up its reserves of mineral with losses exceeding those provided for by its production plan, it reduces its own life and thus upsets overall plans of mineral production.

The waste of some self-ignition minerals may, in certain conditions, lead to underground fires. This applies to most varieties of mineral coal, lignites, chalcopyrites and pyrites.

It thus becomes evident that mineral losses and waste in mining are harmful in many respects and should be minimised as far as possible.

Considerable losses may be permitted only when extracting minerals of little value in which nature is extremely rich (certain building stones, rock salt, etc.), for mining them with reduced losses would lead to a substantial increase in production cost.

## 6. Mines

*Mines* are industrial enterprises whose designation is to exploit or explore mineral deposits.

In the U.S.S.R. mines are in the charge of economic councils.

The operational mining unit is a *mine*, when minerals are obtained by the underground method, and *quarry* (open pit) in the case of open-cut mining.

The mines and open pits are managed directly by trusts, each being in charge of several mines or open pits. The activities of several trusts are guided by *combines* which are directly responsible to the regional economic councils. In some cases the managerial patterns of mining enterprises are somewhat different.

## 7. Terms and Definitions of Mine Workings

Exploitation of mineral deposits involves driving of *mine workings*, that is, formation of excavations in the earth's crust. The process of driving these workings is called *mining*.

Direct removal of useful minerals or barren rock from the earth's crust is called *extraction*.

In the stage of drivage the advancing surface of a mine working is called *face*.

The *face advance* of a working is the distance marking the progress of its face per unit of time.

In shape, size and disposition mine workings present a great variety of patterns.

Let us first acquaint ourselves with mine workings as shown in the figure illustrating their disposition (Fig. 1).

The deposit consists of two minable beds of useful mineral in the shape of shift-faults. The one above is thin, while the one below is of considerable thickness. *Country* bedrocks enclosing the layers lie under relatively thin *overburden*.

A deposit of this type might be mined, for example, by driving the following workings:

1) a vertical mine shaft *1* can be sunk from the ground surface, and crosscuts *2* and *2'* may be driven from it at certain depth to

intersect the beds. From the crosscuts one can drive vertical openings *3* upwards or *4* downwards or else inclined openings *5* upwards and *6* downwards;

2) in the case where the surface is mountainous, instead of extending a vertical shaft and crosscuts it is possible to drive adit *7* from the valley to mine the upper portion of the deposit;

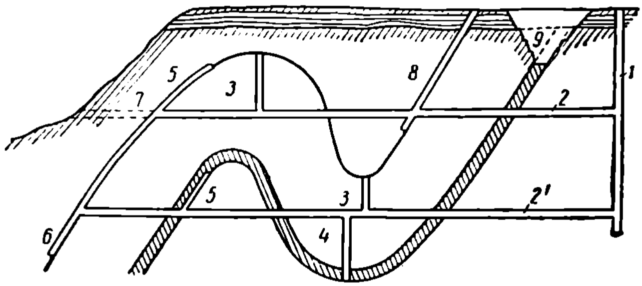



*Fig. 1.* Layout of mine workings

3) the beds can also be worked through inclined shafts *8*, driven from the surface;

4) to a certain depth the thick bed can be mined by open-cut method *9*.

In comparing the workings listed above, one can divide them into two basic groups: *open* and *underground*. The latter are subdivided into *vertical*, *horizontal* and *inclined* openings. In addition to these, there are underground mine workings whose extension is insignificant when compared to their cross-section (usually rather large); these are called *rooms*.

Various *productive workings* are excavated at the sites where the bulk of the mineral is actually mined, that is, in *stopes*.

The initial point of vertical and inclined openings is sometimes termed *mouth* and the terminal—*well* or *sump*.

The lateral sides of horizontal and inclined openings are called *walls*, the lower side—*bottom* or *floor*, and the upper—*back* or *roof*. Surfaces confining vertical openings are called *walls*.

Let us now pass to a more detailed discussion of mining terminology.

### *Vertical Openings*

*Mine shaft* (or briefly *shaft*) is a vertical opening communicating directly with the ground surface and intended to service underground operations. Shafts are classified as main or hoisting and auxiliary, depending on their chief designation. The *main* shaft serves principally for hoisting the mineral to the surface. *Auxiliary* shafts are named after the nature of the principal functions they perform: *upcast fan shaft* (for ventilating mine workings), *drainage* or *pump shaft* (for water disposal), *supply shaft* (for handling mine-fill), etc. Quite often shafts perform several functions at a time and in that case they are called after the principal one.

*Blind* or *dummy shaft* is a vertical opening with no direct communication with the surface. It is provided with mechanical equipment to transport people and materials.

*Test pit* is a small vertical opening sunk from the ground surface for the purpose of prospecting. Sometimes it is also used in exploiting the mine, particularly for purposes of ventilation. In some cases pits are employed for charging explosives in bulk blasting.

*Borehole* is an opening made with the aid of a drill. The diameter of a hole usually comes to a few centimetres or several scores of centimetres. Boreholes with a diameter of around 0.5-2 metres are commonly called *large borings*.

A vertical opening several metres in diameter, driven with the aid of boring rigs, is called shaft.

### *Horizontal Workings*

Generally speaking, these workings are not strictly horizontal. They have a slight slope (of a few millesimals) to facilitate the runoff of water and haulage.

*Adit* is a horizontal underground passage directly communicating with the surface and intended for servicing a mine. Like shafts, adits may be classified as: *main*, *auxiliary*, *haulage*, *drainage*, *ventilating*, etc. A small adit used for exploration purposes is called *prospecting adit*.

*Tunnel* is a horizontal underground passage open to the atmosphere at both ends. In mining the use of tunnels is rather restricted.

*Crosscut* is a horizontal opening that has no communication with the surface and is driven in country rocks at a certain angle to the course or the strike of the rocks. More often than not it is driven across the strike, that is, at a right angle to it.

*Drift* is a horizontal opening that has no communication with the surface and is driven along the strike of a deposit; in horizontally occurring deposits—in any direction. At coal mines drifts are usually called entries.

Drifts or entries can be driven both in the mineral body and in country rocks. In the latter case they are called *lateral drifts*. Drifts play a major part in mining mineral deposits and serve a variety of purposes. Accordingly, different drifts are given special names: haulage or tramming drift, ventilation drift, etc. These, however, are better discussed in the chapter devoted to the description of mining methods and to the terms of some horizontal workings of secondary importance.

### *Inclined Workings*

*Inclined shaft* is an opening similar to an ordinary shaft, but extending from its mouth downward at a certain angle.

*Mine slope* is an inclined excavation that has no immediate communication with the ground level and is used for hoisting minerals and other materials. *Braking incline* is a working of a similar nature, but designated for lowering minerals and other loads with the aid of mechanical devices. *Gravity incline* is an excavation of a like type, but the loads slide down of their own accord, carried on by their weight. Gravity inclines frequently have manways with special ladders. These, however, are not used systematically by people, but serve as access to any part of the incline in effecting repairs and also to help move down jammed materials.

In ore mining the term *raise* is widely used. This is an opening driven to the rise (see Chapter II, Section 2) intended for ventilation, men's passage and transportation of materials. When required, cables and pipelines are laid in raises.

*Manway* is an inclined opening serving principally as a men's passage.

Mining practice also knows *inclined blind shafts, test pits* and *inclined boreholes*, all of them openings similar to vertical workings, but driven at a sloping angle.

### *Service Rooms and Workings*

*Service room* is an underground excavation for mechanical and other equipment, and also for supply and sanitation purposes.

The names given to the numerous service rooms do not require any explanation. These include *underground sheds for electric mine locomotives, main sump storage* or *water chamber, dispatcher's room, storage room for fire-fighting materials* and *underground shed for fire-fighting trains, underground storage room for explosives*, or *powder room, underground first-aid stations* and some others. *Engine* or *machine rooms* are named after the equipment installed: *pump, compressor, transformer* and *slusher rooms, underground electric substation and switch room*, etc.

The term *shaft* or *bottom station* requires explanation. It is an aggregate of underground workings near the shaft designed to service underground operations and to connect the shaft with mainline haulage drifts and ventilation openings.

*Productive workings* are of extremely varied shapes, size and spatial disposition, depending upon the geological structure of a deposit and the mining and stoping methods adopted. Therefore, the definition of their terms is listed in the second part of this book, while those covering open-cut mining are given in Chapter XXV.

## 8. Mine as a Production Unit

As indicated above (Section 6), the *production unit* of a mining enterprise engaged in underground extraction of useful minerals is called a *mine.* It includes all surface structures and the aggregate of underground mine workings equipped for removing valuable minerals and constituting part and parcel of this independent industrial and economic unit of the mining enterprise.

The mineral deposit or part thereof allotted to a mine for working purposes is called *mine field.*

The shape of mine fields depends upon the geological structure of a deposit and may be extremely variable. Projections of mine field boundary lines onto the ground surface are plotted on mine maps and plans.

The size of mine fields is determined by the amount of workable mineral reserves ($Z$) contained therein and by the geological structure of the occurrence. It may vary greatly in most cases from a few hundred metres to several kilometres along the strike of the deposit.

Annual production capacity in tons ($A$) and the overall life ($T$) of each mine are planned in accordance with the established programme for national economic development. It will be clearly seen that the above-mentioned three values are interlinked by the following simple relationships:

$$A = \frac{Z}{T}; \tag{1}$$

$$T = \frac{Z}{A}; \tag{2}$$

$$Z = AT. \tag{3}$$

The above-mentioned relationships are illustrative of the organic connection between annual output and the life of a mine, on the one hand, and the reserves of the mine field, on the other. Proper assessment of these three basic values constitutes one of the major tasks in planning the layout and production scheme of a mine.

It should be noted, however, that $T$ means *estimated* life of the mine, corresponding to the time interval necessary to exhaust mine field reserves, during which the output of the mine $A$ (t) remains steady year in and year out. Actually, however, a certain length of time is required in the initial period of operation to *bring mine production to its planned level*, while in the ultimate years of a mine's life there comes a period of *gradual reduction* in its output, marked by the slowing down of operations. Hence the *overall* life of a mine, from the moment it is commissioned, exceeds the estimated one by approximately 2-4 years.

The mine's annual output $A$, its life $T$ and mine field reserves $Z$ may vary very broadly, depending upon the geological features of a deposit and the type of the mineral. The methods employed for determining these values and the corresponding figures are given in subsequent sections of this book. Here we shall dwell only on some matters of principle.

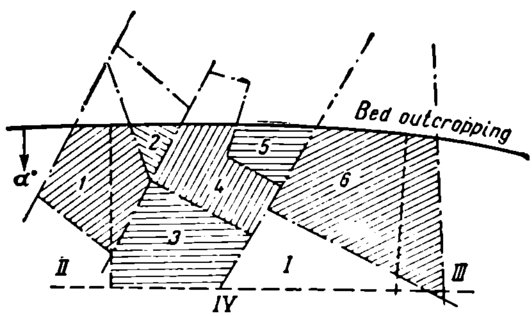



*Fig. 2.* Harmful effect of the boundaries of a private land property on the working of mineral deposits

In capitalist countries, factors inherent in capitalist relationships are of decisive importance in setting annual production rates and fixing the life of a mine. These factors are the drive for maximum profit; private ownership of land, mineral wealth and means of production; limited amounts of capital which the mine owner is able or willing to invest in a given mine; instability of the market demand due to spontaneous changes in the industrial situation, sharp commercial competition, etc.

To mine a deposit under capitalism one has to own or lease a corresponding land plot. The boundaries of private land properties have nothing in common with the geological structure of a mineral deposit, which determines the position of mine workings. For example, the thick line in Fig. 2 points to an outcrop of a mineral deposit, while the thin ones are the boundary lines of private land properties *1*, *2* and *3*. These boundary lines are quite arbitrary in nature and their position with respect to the strike and dip of the bed is absolutely irregular, this giving rise to technical difficulties in exploitation.

Besides, plots *2* and *5* are extremely small. It is not worth while mining them separately; if they are mined, the shafts sunk there will be very small. The larger the mine, the greater capital outlay is required for its construction. It is not always that a private owner is able or willing to invest considerable sums and so it may happen that the capacity and facilities of a mine springing up on a rich deposit are out of all proportion to the reserves contained therein. Finally, in conditions of competition and overproduction common to capitalism, there are frequently overt or secret agreements concluded among businessmen to restrict mineral production in order thus to artificially maintain inflated market prices.

Under the socialist system factors that adversely affect the rational utilisation of natural wealth have been eliminated completely. What we have instead is abolition of private ownership of land, mineral wealth and means of production, and planned national economy.

Therefore, in Soviet conditions the annual production capacity of a mine, its life span and field reserves are determined exclusively from the viewpoint of technical and economic expediency and in the interests of the national economy as a whole. For instance, should technical and economic estimates prove that the deposit shown in Fig. 2 is technically suitable and economically profitable for mining if it is divided into mine fields *I*, *II*, *III* and *IV*, there will be no obstacles to the realisation of such a decision in our country.

Soviet mines are so planned and built as to make the capital invested in their construction and equipment yield the highest possible returns. While in operation, the mine should be run most efficiently and the safety of personnel maintained at a high level. Rational utilisation of capital investments and high labour efficiency make for low production costs. This is achieved through a proper choice of mine structures, complex mechanisation and proper organisation of mining operations.

Mines in their capacity of production units generally are not isolated; there are usually other mines in the vicinity. Therefore, it is possible to simplify and lower the cost of surface plants and mine equipment by erecting buildings and structures that may be commonly used: concentration plants, mechanical and repair shops, well-appointed socialist cities and workers' settlements. Transport lines—railways, highways, tramways, aerial tramways—must be designed so as to offer optimal service to the entire *complex* of adjacent mines. The same holds true for electric power and the supply of drinking, domestic and industrial water.

Hence the major importance of proper organisation in planning the layout of a mine and its operation. It is the basis underlying the entire mine construction work.

CHAPTER II

# OPENING UP OF COAL AND OTHER STRATIFIED DEPOSITS

## 1. Coal Basin and Coal Region

Any coal field representing a continuous formation of coal-bearing strata extending over a specific area in which they have accumulated as the result of a general geological-historical process, may be termed a *coal basin*. A *region* within the boundaries of a coal basin is that portion of the latter the setting apart of which is not only dictated by reasons of administrative and economic nature, but is also determined by the peculiarities of its geological structure. In delimiting such regions, major consideration is given to tectonics and the quality of coal.

By way of illustration we may cite:

a) The central region of the Donets coal basin in the north-western part of the main anticline. Predominant here is the extraction of coking coals.

b) Prokopyevsk-Kiselyovsk region of the Kuznetsk coal basin, which is distinguished by a uniform structure of its coal measures comprising many seams with a multitude of folds with steep wings and abrupt flexures. The types of coal found here differ in property, but some beds include the kinds used for coking.

## 2. Shapes Characteristic of Bedded Deposits

In general terms, the *opening up* of a mineral deposit means driving openings that give access to it from the earth's surface for mining purposes. A more precise definition of "opening up" will be given below (see Section 5).

The external *shape* and spatial disposition of a mineral occurrence is of decisive importance in the choice of the most suitable methods of opening it.

By the nature of their sedimentary origin, *beds* are ordinarily of a *thickness* that is insignificant when compared to the other two dimensions. This thickness is often uniform over quite large areas. Remarkably even in thickness (between 0.5 and 1.5 metres), for instance, are many seams in the Donets basin, this thickness being the same over immense distances (scores of kilometres). A similar phenomenon is sometimes also met within thicker beds. Take, for instance, the Thick Seam in the south-western part of the Kuznetsk basin. Its thickness of about 15 metres remains unchanged over a distance of tens of kilometres. Such phenomena testify to the uniform nature of the accumulation of vegetable matter at the time of the initial formation of coal-bearing strata over quite large areas.

Not infrequently, however, the accumulation of vegetable matter producing carbonaceous substance proceeded unevenly. This resulted in the formation of seams of variable thickness, with *thickenings* ("swells") and *thinnings* occurring in some places, up to and including the complete *pinching out* of individual beds. The local thinnings of seams are called *squeezes*. Frequently alternating accumulations of vegetable matter and mineral sediments (silt, sand and their mixtures) occurring in the process of sedimentation resulted in the appearance of coal beds of *composite* or *multiple* structure interlaid with *bands* of gangue. Strata with frequent changes of thickness and structure are usually called *bedlike occurrences*. Bedlike deposits with irregular horizontal outlines over areas ranging from a fraction of one square kilometre to several square kilometres are a feature peculiar to the Moscow coal basin.

Strata and bedlike deposits seldom retain their quasihorizontal position. An exception to this general rule are the occurrences of the Moscow and Cheremkhovo (Eastern Siberia) coal fields, the stratified manganese ore bodies in the Nikopol area (Ukraine) and some others.

As a rule, however, *tectonic* processes cause the strata to change their horizontal position. Usually they develop *folds* (*plicated dislocations*) or *ruptures* (*disjunctive dislocations*). The principal types of folds are *anticlines* (strata dipping outwards, away from the fold axis) and *synclines* (the form of which is downward concave). Dislocations with ruptures are accompanied by displacements of the disturbed rock in situ. The major types of displacements include *faults* and *shift faults* or *throws*. Quite often one and the same occurrence presents a picture of a folded dislocation and shift faults involving ruptures in the continuity of rocks.

As the result of geological disturbances beds have a dip, that is, a certain inclination to the horizontal, measured by the *angle of dip*. Beds are subdivided into *sloping* or *flat-dipping* (with the angle of dip of 0-25°), *inclined* (a dip of 25-45°) and *steep* (45-90°). The line of maximal inclination in the bed plane is called the *line of dip*.

The horizontal course or bearing in the plane of a bed is called *line of strike*. It is customary to refer to the direction of this line simply as *strike*. The horizontal position of the line of strike is determined by its *azimuth*, that is, by the angle between the plane of the meridian and the line of strike. The lines of dip and strike are normal to each other.

The thickness, angle of dip and the strike of a bed are referred to as *elements of occurrence.* Beds in areas where the elements of occurrence are characterised by constant values are called *regular* or *uniform*; in other words, *undisturbed.*

Because of irregular sedimentation at the time of bed formation, i. e., due to *genetic* causes, and also because of subsequent tectonic disruptions, the shapes and elements of occurrence peculiar to stratified deposits may assume variable and often very complex form.

The planning of mining operations requires graphic representation of shapes and elements specific to the occurrence of any given mineral deposit. The method used for this purpose is that of *isolines.* Its chief features may be summarised as follows.

The relief of the earth's surface is usually represented by *contours*, or *isohypsometric lines*, that is, lines of equal elevation. These can be obtained by cutting the ground surface with imaginary equidistant horizontal planes (for example, every 10, 1, 0.5 m). Each one of these contour lines has its own elevation (bench) mark. A contoured topographic map or plan gives a clear-cut idea of all the features peculiar to the ground surface relief and is indispensable for planning and designing ways of communication, water works and other installations.

Quite similarly to this, isolines (that is, lines of equal properties) may be employed to depict features specific to the occurrence of a mineral deposit.

The *isolines of a bed bottom* are similar to contours on topographic maps, but they follow the bottom of a seam to demonstrate its geometric characteristics. In the same manner one can construct *isolines of the roof* of a deposit, etc.

Variations in the thickness of a bed may be illustrated quite clearly by *isolines of equal thickness*, connecting points of identical thickness in a stratum and drawn at regular intervals (for example, every 0.2 m).

Isolines may be used to represent not only geometrical but other characteristics and features of a mineral deposit as well. They may, for instance, depict the percentage of ash or volatile matter in coals, etc.

While stations needed to plot contours on topographic maps are secured by levelling, those required to construct isolines characterising the specific properties of a mineral deposit are furnished by boreholes in the course of prospecting and by underground workings in the course of mining. The richer the initial data supplied in the form of stations, the more reliable the plotting of isolines and the more detailed is the picture they present of the mineral occurrence as a whole.

To prepare plans for the opening up and initial development of mineral deposits, particularly those of a more complex structure, it is indispensable to plot isolines on special maps.

A Russian scientist, P. Sobolevsky, gave much of his time and effort to working out the theory of representing in isolines the various important characteristics of mineral deposits. It was he too who propagated practical utilisation of this remarkable method.

## 3. Mine Field

In Chapter I mine field was defined as a whole deposit or the portion of it allotted to be worked by a mine. The plans below show its *projection* on the ground surface.

The form or outline of a mine field depends in the main upon the features characterising the occurrence of a deposit.

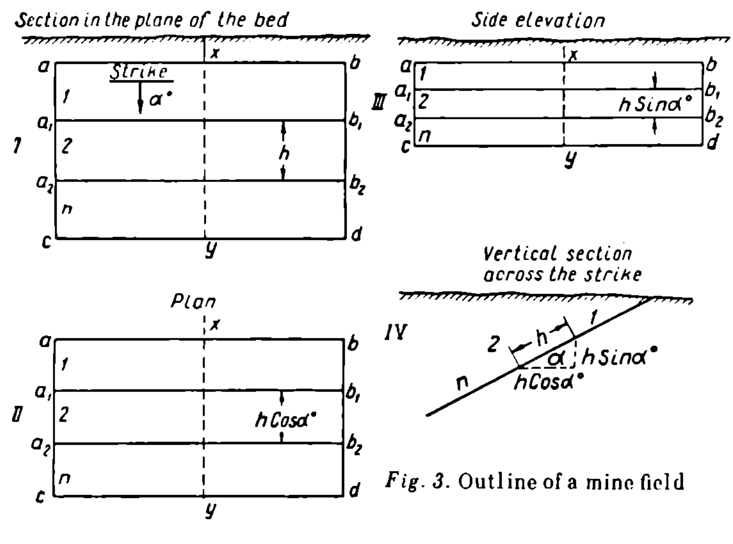



*Fig. 3.* Outline of a mine field

In bedded deposits with regular elements of occurrence, the mine field, whenever possible, is given the shape of a rectangle extending along the line of strike. Fig. 3 shows an outline of a single seam mine field drawn in the plane of the seam (*I*); its plan, that is, projection on the horizontal plane (*II*), longitudinal side elevation along the strike (*III*) and vertical section or side elevation across the line of strike (IV).

The upper *ab* and the lower *cd* boundary lines of the mine field run along the strike of the seam, while its lateral lines *ac* and *bd* — down the dip.

The upper mine field boundary is also called *up-dip limit*, the lower one—*down-dip limit* and the lateral boundaries—*strike limits*.

In underground mine plans and maps mine fields with their workings are usually shown in horizontal projections (*II*).

In steeply dipping deposits these are supplemented by side elevations (*III*). In plans and side elevations the dimensions of the lines drawn along the strike alone remain undistorted. Therefore, in preparing layouts, the drawings of the mine field (or part thereof) are sometimes made in the plane of the seam (*I*), where all the dimensions of the field retain their true values. The schemes of the mine field given above show vertical *cross-sections* representing the typical areas of the deposit.

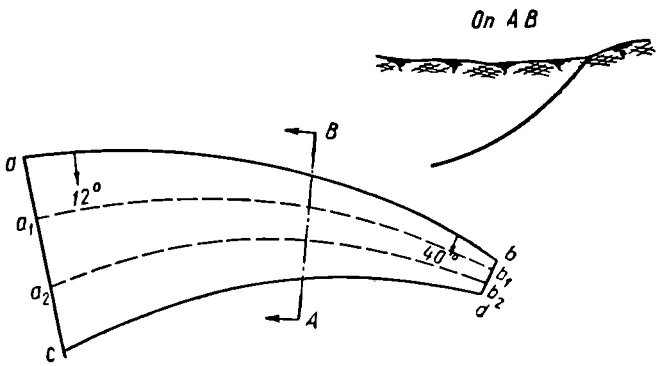



*Fig. 4.* Mine field extending over a fold of a deposit

An irregular occurrence of the deposit modifies the shape of the mine field. Fig. 4 shows an outline plan of a mine field extending over a fold of the deposit. The upper boundary *ab* lies near the *outcrop* of the curved seam. The angle of dip of the seam is also subject to changes—from 12° to 40° at the site of exposure, with the occurrence gradually *flattening* at greater depth. The lower boundary *cd* of the field is level, running along the strike. Dash lines show the structural contours of the seam floor.

Fig. 5 represents a plan of a mine field whose lateral boundaries *ac* and *bd* are determined by the position of large faults extending obliquely (diagonally) to the strike of the seam.

Fig. 6 depicts a plan of a mine field whose boundaries are determined by the contour lines of sheetlike deposit. Here too dash lines represent the surface contours of the seam bottom.

The size of mine fields varies greatly, this depending upon the reserves of the useful mineral they contain, the number and thickness of coal seams, the depth of mining, as well as annual output and the service life of the mines. The extension of a field along the line of strike ranges from a few hundred metres for small mines to several kilometres for larger ones. The basic points of the analytical method used for determining an economically expedient size of a mine field are discussed in Section 14 of this chapter.

The reserves of a mine field ordinarily are worked out for several decades, most often for 10-40 years (see Section 15). The portion of a mine field where the mineral has been extracted is called *mined-out area* or *space*. If the mined-out area is no longer worked and is abandoned as useless, it is called *old* or *abandoned working*.

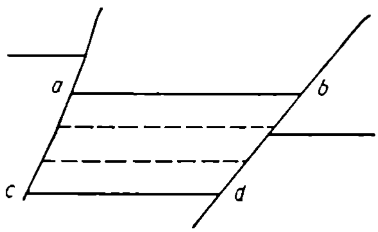



*Fig. 5.* Mine field with boundaries determined by faults

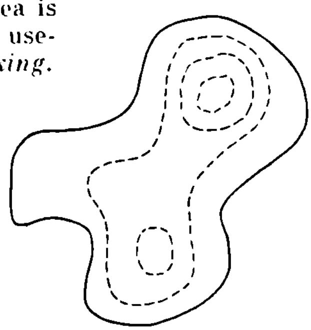


*Fig. 6.* Mine field in a sheetlike deposit

*Maintenance* means upkeep in proper condition of a mine working, its timber and tracks. Discontinuation of the maintenance and employment of a mine working is termed *abandonment* or *closure*.

Reliable communications between underground excavations and the ground surface require the provision of *no less than two exits* from each mine field, each suitable for the passage of people. The availability of no less than two exit openings is also needed for ventilation purposes. These exit openings should be no less than 30 metres from each other, and when shaft houses are built from fire-proof materials—no less than 20 metres. The miners should know all the exit openings from underground workings.

## 4. Division of a Mine Field into Levels

A mine field with sufficient coal reserves for 10-40 years is worked gradually over this period, by separate sections—*levels* or *panels*.

Since, with few exceptions, deposits tend to occur at a certain angle of pitch the mine field is worked by sections extending along the strike and, consequently, separated from one another by horizontal lines (see Fig. 3). These portions of the mine field are called *level intervals*. The relation between the height of the level interval and annual mine output is discussed below. When the values of the factors characterising the occurrence of a deposit are constant, that is, when its position is in one plane, the level intervals assume the form of rectangles (see Fig. 3). When the strike or the angle of dip of the deposit within the limits of the mine field is subject to variations, the configuration of the level intervals become curvilinear (see Fig. 4).

To provide for the haulage of the broken mineral, men's passage and ventilating air currents along the boundaries of level intervals, level drifts or entries ($ab$, $a_1b_1$... in Fig. 3) are arranged. As mining operations in any given level interval progress, these drifts are extended along the entire mine field to its boundaries.

*A level interval is thus a section of the mine field extending along the strike of the deposit, bounded on the side of the rise of the seam and its dip by drifts or entries driven along the entire length of the mine field, and in the direction of strike—by the mine field boundaries.*

Drifts confining a level interval are called *level* or *main* drifts. Since the lower level drift in any level interval is used largely for transporting the broken mineral, it is often referred to as *main haulage* (tramming) drift or entry. The principal function of the upper level drift is to provide passage for return air and for this reason it is usually called *ventilating entry* or *airway*.

In Figs. 3 and 4, $abb_1a_1$ and $a_1b_1b_2a_2$ are level intervals; $a_1b_1$—the main haulage drift or entry and $ab$—the ventilating $abb_1a_1$ level drift.

Separation (delimitation) of level intervals by drifts makes transportation convenient, since they remain horizontal whatever the angle of dip of the deposit may be. But this point is of no importance in mining horizontally occurring deposits, for, irrespective of the direction in which any working is driven, it always remains level. That is why the mine fields of horizontal or low dipping deposits are divided into so-called *panels* and not level intervals (see Section 10).

The *inclined height of level interval h* is measured along the line of dip (see Fig. 3). With the angle of dip equalling $\alpha$, the corresponding *vertical* or *true height* of the level interval is equal to $h \sin \alpha$.

The useful mineral obtained in the faces is transported through level haulage drifts and other underground workings to vertical or inclined shafts, by which it is hoisted to the surface.

When the mine field is more or less regular in shape and the distribution of the mineral contained therein is uniform, hoisting shafts are sited in the centre of the mine field strike, that is, on the line $xy$ in Fig. 3. This, as we shall see later (Section 14), helps to economise on underground tramming and also to reduce other expenditure. In this case, the mine field is divided into two equal wings or flanks. Generally speaking, the term *wing* denotes that *portion of the mine field which lies on either side of the main hoisting shaft.* Accordingly, each level, as a rule, has two wings. The wings of a level or of a mine field are usually called cardinal points (for example, eastern wing or flank, etc.). Mine fields or levels with flanks of equal size are generally called *equilateral*, or conversely—*nonequilateral*.

A mine field or a level is called *unilateral* when the main hoisting shaft is located near one of its lateral boundaries.

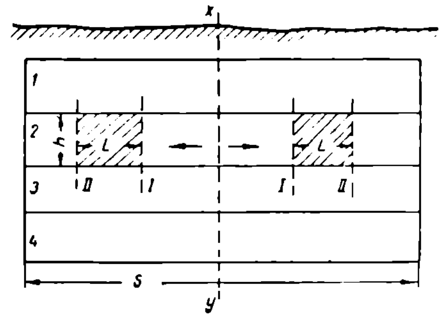


*Fig. 7.* A diagram for estimating level intervals

It is desirable to have such inclined height $h$ that the working of only *one* level at a time will suffice to extract the planned annual amount of the mineral through shaft $A$, since this will maximally simplify mining operations.

Let us select interval $h$ between levels (m) for a mine field of regular shape and uniform distribution of coal reserves, knowing that, to ensure the annual planned output of shaft $A$ (tons), it will suffice to work one level at a time.

Let us assume, for example, that it is the second level that is being worked (Fig. 7) in the mine field and at a certain moment the working stopes follow schematically lines $I$, $I$. Let us, then, suppose that in a year's time these stopes have advanced distance $L$ (m) to positions $II$, $II$. That will mean mining area $hL$ on each flank, or a total of $2hL$ sq m. If, taking into account the thickness and structure of the seam, the coal yield from 1 sq m of its area averages $p$ (t) ($p$ is the *productivity* or output of coal per 1 sq m of the seam), then are $2hL$ would include the known or *blocked-out* reserves of coal—$2hLp$ (t). These reserves cannot possibly be removed fully, since the extraction of coal within a level entails *operational* or *exploitation* losses (wastage); in other words, *coefficient of recovery* $c$ in the level is below 1. Therefore, the annual tonnage of reserves extracted within the bounds of each wing will come to $2hLpc$ (t).

Thus, with only one level being worked at a time in a mine field, the annual output of mine $A$ will be

$$A = 2hLpc. \tag{1}$$

This simple equation establishes a relation between the annual output of mine $A$ and level interval $h$, annual advance of stopes $L$, average productivity per 1 sq m of seam $p$ and coefficient of recovery $c$.

With the aid of formula (1) we may determine the slope distance between levels as follows:

$$h = \frac{A}{2Lpc}. \quad (2)$$

Thus, the higher the annual output of the mine, the greater the slope distance between levels becomes; and it becomes the smaller, the greater the annual advance of working stopes, productivity of the seam and the coefficient of coal recovery.

Numerical values of the entities forming equations (1) and (2) need some explanation.

The annual advance of stoping operations is closely related to the method used in mining a seam. This will be discussed in detail later on. It is important, however, to stress the fact that, in making use of the formulas (1) and (2), one should *take factually attainable average annual advance of stoping operations in a wing*, that is, to reckon with possible stoppages at some faces and the need to have spare stopes. Consequently, *in estimating the level interval, one should already have a clear idea of what mining method to employ in working the deposit.*

In the case of the regularly occurring measures in the mines of the Donets basin, the annual advance of stoping with a flat pitch comes to 300-400 metres; with a heavy pitch— to 400-500 metres.

Actual fulfilment and overfulfilment of the planned annual advance of a mining operation is of a paramount importance. From formula (1) it follows that should the planned advance $L$ be not achieved, the extracted tonnage $A$ will fall short of the planned figures and, consequently, the annual production schedule of the mine will be upset.

The following are the reasons which make it necessary to provide for spare faces in determining the annual advance of mining operations $L$.

The summary line of all mine faces (working and development) is called *total* stope footage. This includes the footage of both *active* and *spare* faces.

Consequently, the total advance of stoping operations is determined not only by the progress of active faces, but also by the availability of spare ones, and the average advance of mining operations must be established with due consideration of this fact.

The stope footage is often referred to as breast *front*. Fulfilment of a mine's annual production programme requires a breast front of adequate extent. Its reduction would mean a corresponding drop in mined tonnage, if the rate of advance of active faces remains the same. Hence the immensely harmful effect of the total and active stope footage reduction to a level below the plan figures.

On the other hand, an excessive breast front, that is, one surpassing the requirements of the production programme, is likewise unprofitable, for this would mean driving and maintaining unnecessarily large footages of mine workings.

Failure to drive development openings in due time will inevitably result in the reduction of the necessary breast front.

The output per 1 sq m of seam $p$ (t) depends directly upon its thickness and structure, that is, on the availability or absence of gangue bands or intercalations, and on the unit weight of coal. The latter is determined during the exploration of the deposit. In preliminary estimates made in planning mine operation, the volume weight per 1 cu m of coal in place (its density) may be assumed to be 1.5 metric tons for anthracite, 1.3-1.4 tons for mineral coal and 1.2 tons for lignite. For instance, if a seam is 1.2 metres thick and is devoid of gangue bands, the yield of coal per 1 sq m, with its unit weight equalling 1.3 tons, will be $p=1.2\times1.3=1.56$ tons. In seams intersected by bands of barren rock, the minable thickness of a seam is computed by subtracting the height of bands from its *aggregate* thickness and the resultant figure is used in calculating the output per 1 sq m of the seam.

*Operation* or *exploitation losses* of coal usually range between 10 and 15 per cent, but in working regularly occurring thin seams they drop to 3-5 per cent. However, with inadequately conducted mining operations, particularly in working thick seams by caving methods, these losses may increase considerably.

A detailed explanation of the harmful effects caused by the unduly high wastage of useful minerals during their winning will be given later. Meanwhile, we shall only emphasise the following point. The greater the losses are, the lower the coefficient of recovery of coal reserves $c$. Equation (1) shows that with the same level interval $h$, equal coal yields $p$ and annual advance of mining operations $L$, but with percentage losses higher than envisaged by the plan, the annual production capacity of mine $A$ will drop. Conversely, with all other conditions being equal, a seam worked with losses below those planned opens up good possibilities for the overfulfilment of the mine's production programme.

Equation (1) reveals that the annual mine output is directly proportional to the level interval. With a bed occurring at a gentle or low dip, the level interval may reach hundreds of metres; a steeper dip, on the other hand, narrows the range of this interval.

Formulas (1) and (2) are drawn up for the desirable occasion when the mine production programme can be assured by the exploitation of one single level at a time. To increase mine output stoping is sometimes effected on two and more levels. That, however, complicates the underground haulage layout and mine ventilation, increases the footage of mine openings to be maintained and, finally, impairs the efficiency of mine workers. For this reason restriction of mining to one level at a time should be considered a normal practice. The next level must be developed beforehand. At the time when stoping operations *slow down* in one level and work is *deployed* in another, the breaking of mineral is continued in two levels simultaneously and to the extent sufficient to implement the mine's production programme.

Below are two practical examples illustrating the use of formulas (1) and (2).

*Example 1.* Determine the annual output of a mine if it works a single low-dipping coal seam with an average yield of 1.5 metric tons of coal per 1 sq m. The level interval is 300 metres, operation losses—6 per cent and the annual advance of stoping in one wing—400 metres.

Formula (1) gives us

$$A=2\times 300\times 400\times 1.5\times 0.94\simeq 340{,}000 \text{ tons.}$$

*Example 2.* Determine the distance between levels sufficient to ensure mine output of 500,000 tons of coal a year if the mine works a single coal seam yielding $p=2$ metric tons, with the annual advance of stoping being 450 metres and operation losses coming to 10 per cent.

Formula (2) shows that the slope distance between the levels is

$$h=\frac{500{,}000}{2\times 450\times 2\times 0.9}=327 \text{ metres.}$$

## 5. Opening and Developing a Mine Field

In Chapter I the opening of a deposit was preliminarily defined as driving workings to give access to a mineral occurrence it is planned to exploit.

When a mine field is worked levelwise, the term *opening* may be given a more precise definition.

The deployment of stoping in a mine field necessitates driving openings of two types:

1) workings indispensable to start driving level entries in the mine field. Let us call excavation of such workings *opening* and the workings themselves—*early development* or *permanent mine openings*;

2) mine workings driven within the limits of one level. Their purpose is to "prepare" the level for stoping operations and ensure access to working faces. Such excavations are called *development openings* and their driving—*development work* or, briefly, *development* for stoping.

It should be pointed out, however, that sometimes mine fields are not divided into levels, but into so-called "panels" (Section 10) and in this instance the definition of "opening" and "development" cannot possibly be associated with the concept of "level".

Accordingly, in a wider sense, *the opening of a mine field means driving of workings that give access to the mine field from the ground surface and ensure the driving of development mine workings, and development is the driving of mine openings that make it possible to conduct stoping operations.*

Considering that we have defined the concepts "opening", "development" and "stoping", it would be quite appropriate to give a more detailed definition of the term "mining" which we have already used on many occasions: *mining* a deposit implies an aggregate of opening, development and stoping operations conducted in set sequence.

Opening is effected largely through inclined and vertical shafts, vertical or inclined shafts and crosscuts, adits (Chapter IV, Section 7), and by combining these methods.

Let us review all the above-cited methods of opening, first for individual (single) seams and then for their series.

## 6. Opening by Inclined Shafts

Opening of seams through inclined shafts is one of the simplest methods of early development (Fig. 8).

From the ground surface inclined shaft *ab* is sunk through the coal seam to the lower boundary of the first level, and from this shaft level strike entries can be driven. These allow development openings to be made in accordance with the mine layout schedule and opening up of stopes shown schematically in Fig. 8 by lines *cd* and *c'd'*.

The coal broken down at the faces is transported along the main entries to the shaft station and then brought to the surface through the inclined hoisting shaft.

When the bulk of the mine output is handled through a given shaft station, this station is said to be at "production level". In other words, *production level* is the haulage level in which stoping is largely done. The levels are either numbered in regular order from the surface or designated by their actual elevation below the top of a shaft or sea level.

The strike levels *cc'* and *dd'* are protected from the pressure of rocks settling over mined-out spaces either by coal pillars or packwalls (rib fills) laid out along the entries (Fig. 8).

Since rib fills and coal pillars reduce the rock pressure without eliminating it altogether, the entries must be *maintained* in good repair.

Fig. 8 is illustrative of an instance when the first stoping faces are started near the hoisting shaft and the general direction of stoping is from the shaft towards the outer boundaries of the mine field. This order of extraction within the level (or in the whole of the mine field) is called *advance mining*. There is also *retreat mining*, in which level strike entries are pushed forward over the entire length of the mine field prior to stoping. The first stoping faces are started near the outer boundaries of the field and the general direction of stoping is from the boundaries of the field to the hoisting shaft. Advance and retreat mining is compared in detail below (Section 13).

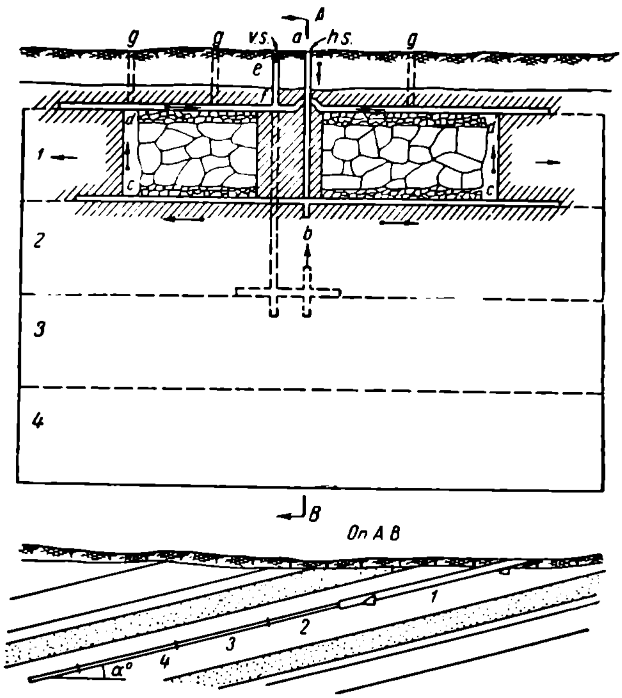



*Fig. 8.* Opening up by inclined shafts

To aerate underground workings in the mine field, a *ventilating* shaft is sunk in addition to the main or hoisting shaft. If the hoisting shaft is inclined, the ventilating shaft is inclined too.

For ventilation proper it suffices to sink an air shaft only to the upper (ventilating) entry of the given level. For instance, in mining the first level (see Fig. 8) its length may be limited to *ef*.

An *exhaust* or *suction* fan is installed over the ventilating shaft on the ground surface to rarefy (*depress*) the air in the underground workings of the mine field. As the result of this depression atmospheric air enters the mine through the main shaft and the overall scheme of air flow is as follows: the *down-cast* current of fresh air proceeds along the main shaft, branches out to haulage entries of the both level flanks, *rises* and sweeps sloping faces, moves further on along the airways or ventilating entries and is then *cast up* as a return current to the ground surface through the ventilating shaft. Since one of the air currents (in Fig. 8 that of the right wing) by-passes the hoisting shaft, an *air* bridge is set up at this point.

Besides being service openings, the hoisting and ventilating shafts also serve as exits from the underground workings to the surface.

As already said, for adequate aeration it suffices to sink the ventilating shaft down to the upper boundary of the level mined at the time. But, since this shaft also serves many other purposes as a passage for men, auxiliary hoisting installation, and for laying water and compressed-air pipes and electric cables, it is generally sunk to the level of a hoisting shaft.

Finally, it is necessary periodically to *deepen* the main production shaft to switch over to mining the next level. To facilitate this deepening process, the air shaft is sometimes driven one level deeper (dash line in Fig. 8). This makes it possible to excavate in advance the necessary service rooms and workings near the shaft on the level below, that is, in its main haulage entry, and to deepen the hoisting shaft by *raising* it, as outlined in Fig. 8. This method of main-shaft sinking through a previously driven ventilating shaft allows us to reduce to the minimum the time necessary for moving hoisting operations from one level to another. Sinking by raising requires very accurate surveying to determine the direction of mine openings in order to secure their adequate connection.

The described ventilation scheme of the mine field involves setting up one main fan at a short distance from the mouth of the hoisting shaft (central ventilation scheme).

But in mining the level nearest to the ground surface, when the upper boundary line of the mine field is not deep, it is possible to use yet another method of aeration. After a certain distance (several hundred metres), the upper airway or ventilating entry is connected with the ground surface through pits *g* (dash line in Fig. 8). These pits serve as openings for men to communicate with the underground workings and for the delivery of timber and other supplies, as well as for installing small fans over them, instead of one main fan. By reducing the extent of air currents, this method does away with the maintenance of the ventilating entry over its entire length and limits it only to the sections between individual pits, thus cutting down the expenditure on repairs of this opening. To decrease the length of pits *g* it is permitted to sink them vertically.

This method has its shortcomings, for it necessitates shifting fans from place to place. In this connection there is yet another scheme possible: the main fan is set up at the hoisting shaft, but operates as a *blowing unit*, producing *compression* of air in underground workings.

The contaminated return air is cast through the pits over which no fans need then be installed.

Thus, the opening up of a mine field through inclined shafts requires driving two parallel shafts—hoisting and ventilating. In bigger mines a third shaft is sometimes sunk to meet auxiliary needs. To protect shafts from rock pressure solid blocks of coal—*shaft pillars*—are left near them (see Chapter XXIII). To facilitate ventilation during the shaft-sinking operations, through-cuts are driven in the pillars, which are subsequently equipped with a bulkhead to separate the intake and return air currents.

In the early development of a mine field it is the uppermost level that is the first to be mined, followed by the ones below it, in the sequence of 1, 2, 3, this being referred to as *descending order* of level mining. The opening and development of each subsequent level must be started well in advance. The time-schedule for these operations should be so compiled as to provide for a considerable time margin over that envisaged by planned estimates.

In inclined shafts coal may be hoisted by different vehicles—mine cars, skips and conveyers. Hoisting by mine car is done by using endless or tail-rope systems. With a high dip mine cars can be put on special flat carriages (Fig. 9*a*). One of the greatest shortcomings of hoisting by mine car is, however, the need to employ a large staff for servicing the hoisting plant and its low efficiency. Skip and conveyer methods of hoisting are much more effective (Figs, 9*b* and 10). Their operation can be automated to a very considerable extent.

Hoisting plants with belt conveyers, however, can be used only when the inclination of a shaft does not exceed 18-20°. Fig. 10 illustrates a conveyer hoisting plant at the S. M. Kirov Mine in the Cheremkhovo coal fields (Eastern Siberia). Underground, coal is dropped from mine cars *1* into a small pocket with a feeder, from which it is fed regularly to belt conveyer *2*; on the surface, it can be unloaded directly into charging hoppers *3* or discharged at dump *4*.

Air shafts or slope manways, driven parallel to the hoisting shaft, usually serve as exits from the underground workings of a mine field opened through inclined shafts. If two inclined shafts are utilised as exit openings, one of them must be equipped with mechanised plant for man-hoisting. To provide for safe exit in case the mechanical hoisting plant breaks down, a shaft with a track gradient of 7-15° should be provided with railings; if its gradient ranges between 15 and 30°, with railings and gangboard; if the slope is between 30 and 45°, with staircases and railings; if the slope of the opening exceeds 45° there must be a special staircase with resting places.

In openings with a slope of up to 30° transportation of men is permitted only in special mine cars provided with overhead cover.

If the track gradient exceeds 30°, the men are also carried in special mine cars or cages.

Opening a deposit through inclined shafts has its advantages and drawbacks.

The former include:

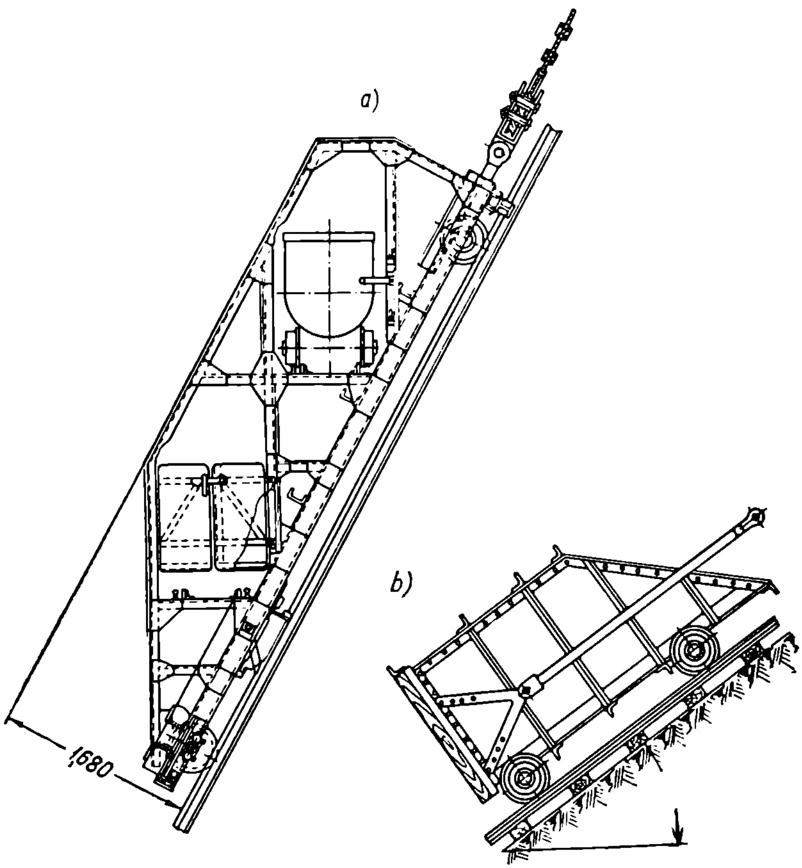



*Fig. 9.* Cage and skip for inclined shaft hoisting

1) with the inclined shaft driven in the mineral, additional information may be obtained on the deposit, complementing that gained at the time of its detailed exploration. This may include data on the structure and features of the mineral bed and the wall rocks; in particular, it becomes possible to determine more accurately the range of the weathering zone and the depth at which the coal is

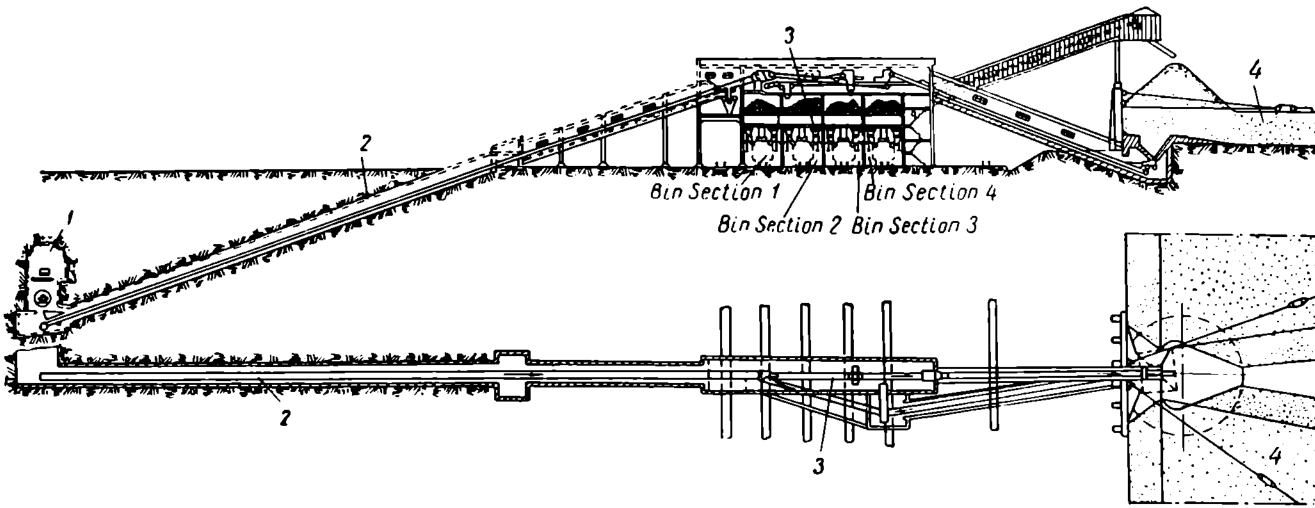



*Fig. 10.* Conveyer hoisting in an inclined shaft

workable; in the case of coking coal beds, it makes it possible to ascertain the depth from which coal is suitable for coking;

2) the coal extracted during shaft sinking may be put to immediate use;

3) an inclined shaft driven in a coal seam does not intersect cap rocks, which may include aquifiers which complicates the driving of mine openings;

4) the cost of driving 1 metre of inclined shafts, these being openings excavated in a coal bed, is lower than that done in country rocks only;

5) as pointed out before, inclined shafts with an angle of slope not exceeding 18-20° make it possible to use highly efficient belt-conveyer hoisting plants.

The disadvantages of inclined shafts include:

1) greater length than that of vertical shafts sunk to the same depth;

2) increased length of hoisting and greater wear of hoisting ropes, compared to that in vertical shafts;

3) lower permissible hoisting speed than in vertical shafts, this reducing hoisting efficiency; with inclined skip hoisting this drawback becomes less pronounced;

4) greater outlays for the maintenance of inclined shafts, since rock pressure in comparable conditions makes itself felt more in inclined openings than in vertical ones.

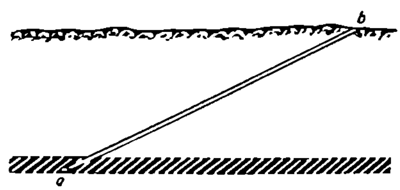


*Fig. 11.* Opening through an inclined shaft driven in country rock

A number of conditions are essential to justify economically the opening up of a deposit through inclined shafts and even to make it technically possible.

1. If, as is customary, inclined shafts are driven in a mineral bed, the overburden covering the bed outcrop should be of but slight thickness.

A flat or very gentle dip with a smooth relief of the ground surface makes sinking of inclined shafts quite impossible. In these conditions, however, inclined shafts may be driven in *country rocks* (Fig. 11). This method, for instance, was employed in opening an extensive field at the Kirov Mine in Cheremkhovo coal fields (Eastern Siberia). Coal there is hoisted by a powerful belt conveyer set at an angle of 18°. Sinking and operating inclined shafts in steeply dipping beds is difficult (as regards repairs and hoisting conditions), and they are very seldom used in mining coal deposits.

2. When the rock topping the bed outcrop is extremely aquiferous or running, inclined shafts are ruled out, since the driving of inclined openings in such conditions is a very difficult task.

3. One prerequisite for driving inclined shafts in a mineral bed is its regular occurrence, that is, absence of faults and steep folds.

4. It is impermissible to sink inclined shafts in thick coal beds, particularly in those with self-igniting coal, for a fire due to fissures caused by rock pressure developing in coal pillars adjacent to the shaft may have very serious consequences. Furthermore, shaft sinking in thick beds entails heavy losses of coal in shaft pillars.

In favourable conditions, opening through inclined shafts is practised quite frequently, particularly in the case of mines with low and medium production capacity.

## 7. Opening up Through Vertical Shafts

Let us examine and compare three possible locations for a main vertical shaft in relation to the mine field (Fig. 12): *I*—the shaft is sunk to the upper boundary of the mine field, *II*—to its lower boundary, and *III*—somewhere in the centre of the field.

In the case of location *I*, the depth of the vertical shaft is minimal, but early development of levels necessitates the arrangement of a *permanent incline*, which gradually extends to the length of the mine field along its strike. In general, this method is similar to that of opening a deposit through inclined shafts with all its inherent shortcomings and is, moreover, complicated by the presence of a vertical shaft.

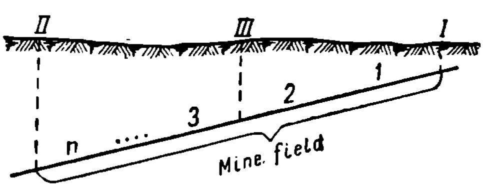



*Fig. 12.* Three possible locations of the main vertical shaft with respect to the mine field

The obvious disadvantage of location *II* is the maximal depth of the vertical shaft. To pass coal from the upper levels down to the shaft, it is necessary to drive a long *permanent slope* along the entire mine field (with the exception of the uppermost level). The broken-down coal must then be passed through it over a great distance.

If the vertical shaft is sunk at some point *III*, the upper portion of the mine field (*up-dip field*) can be mined through a permanent slope and the lower portion (*down-dip field*) through a permanent incline. In this case, the length of both the permanent slope and incline will be moderate compared to that of locations *I* and *II* of the vertical shaft. Hence location *III* is held to be the most propitious.

The opening up of deposits through vertical shafts requires a rather complicated general ventilation scheme and, therefore, we shall dwell on it in greater detail. As an example, let us take a mine field with

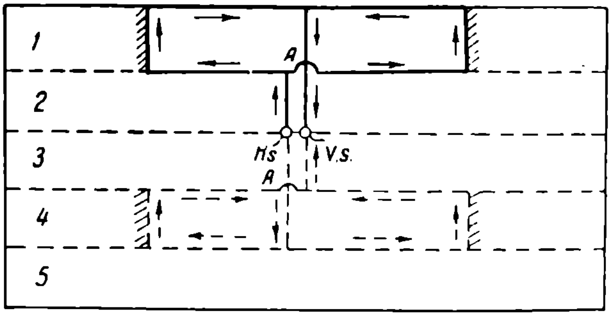



*Fig. 13.* Ventilation scheme for a mine field with the hoisting and ventilating shafts in the centre

five levels—two in the up-dip field and three in the down-dip one (Fig. 13).

In its relation to the main hoisting shaft the ventilating shaft may be sited in a variety of ways. The most common location is *central*, when the two shafts are sunk side by side (Fig. 13). In this case, for example, during the mining of the first level (see arrows in Fig. 13) the fresh-air current enters the mine through the hoisting shaft, rises to the active level via an uphill opening (permanent slope or its manway), branches out to both level flanks along the level strike entries, then sweeps stoping faces as it ascends, passes along the ventilating entries or airways and, uniting into a single common current, goes down the incline and, finally, returns to the ground surface through the upcast shaft.

An analogous scheme for ventilating the workings of one of the levels, for instance, the fourth in the down-dip field, is depicted in the same drawing by dash arrows.

At the sites of air-current intersections (points *A* in Fig. 13) *air-bridges* are arranged. The scheme of their arrangement is illustrated in Fig. 14. If the airway is provided with transport equipment (conveyer, mine tracks), it must be straight at the site of the air-current intersection and a bent entry runs under it (*I*). In the opposite case, the strike entry runs straight and the bent airway lies above it (*II*).

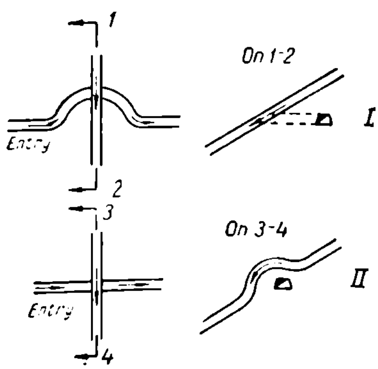



*Fig. 14.* Diagram showing the arrangement of air bridges

If the ventilating shaft is sunk *near the upper boundary of* *the mine field,* the ventilation scheme will be as shown in Fig. 15.

Finally, the air shafts, and there must be two of them in this case, may hold *flanking* positions, located near the lateral boundaries of the mine field (Fig. 16). Since in this instance air currents circulate from the central part of the field, where the downcast hoisting shaft is sunk, and travel towards the upper corners of the field, this ventilation scheme is called *diagonal.* Under it the air currents travel as follows. The fresh air current passes down the hoisting shaft and, during the mining of the up-dip portion of the mine field, rises along

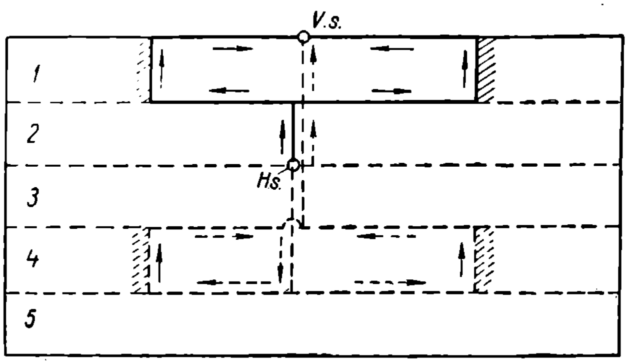



*Fig. 15.* Ventilation scheme with the upcast shaft located near the upper boundary of the mine field

the uphill opening, where it splits into two separate currents. These move towards the flanking shafts, sweeping the working faces on their way. Consequently, in this instance, there is only a uniflow or *straight-way* movement of the air currents, without their moving in the direction opposite to the initial one, as is the case when the shafts are located in the centre or when the ventilating shaft is sited near the upper boundary of the mine field. To make diagonal or unidirectional ventilation of the levels in the down-dip field possible, it is necessary to drive and maintain extensive inclined openings for return air currents at the lateral boundaries of the mine field (see dash line in Fig. 16).

Let us compare the advantages and shortcomings of the three above-cited modes of locating hoisting and ventilating shafts.

At big mines, a ventilating shaft is utilised for a score of needs apart from mine aeration. It is used to accommodate an auxiliary hoisting plant, is equipped with a ladder way, is utilised for laying drainage pipelines, electric cables and compressed air pipes, etc. Because of this, to achieve maximum concentration of the equipment, it is best to sink the ventilating shaft alongside the hoisting one, that is, centrally. Another great advantage of this layout is the fact that, during the construction of the mine after the sinking of shafts, these shafts may be rapidly connected by *through-cuts* ensuring two exits to the ground surface and facilitating normal ventilation of the underground workings. The nearby ventilating shaft may be used to raise the hoisting shaft if this becomes necessary. That is why, in building big mines, preference is given as a rule to the central location of shafts, although it has its disadvantage: variable length of the main air currents and, consequently, irregular operation of mine fans.

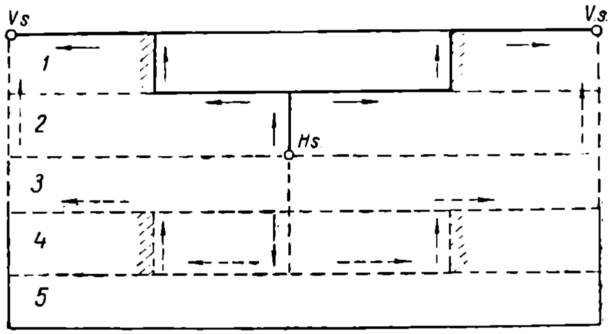



*Fig. 16.* Diagrams explaining diagonal ventilation in a mine field

Although the length of the ventilating shaft sunk near the upper boundary of the mine field is minimal, central shaft location in this case offers no advantages. That is why ventilating shafts are so located only in small mines.

From the standpoint of ventilation, the diagonal scheme (see Fig. 16) has great advantages: 1) the fans operate uniformly inasmuch as the length of air currents passing along the strike entries is constant; 2) when the fan fails in one wing of the field, underground workings are kept aerated to a certain degree by the other.

On the other hand, adoption of the diagonal ventilation scheme necessitates sinking and equipping two ventilating shafts and requires the preliminary driving of extremely extensive through-cuts connecting the hoisting and ventilating shafts. Therefore, this scheme is not planned for the initial stage of mining. But if, as it will be shown below, there are other, formerly worked-out, fields lying over the one in question, old shafts and other mine workings may be used for the realisation of this scheme.

As we see, to open up a deposit through vertical shafts, the mine field is divided into an up-dip portion, in which the levels are opened through permanent mine slopes, and a down-dip one, where they are developed through mine inclines.

The following considerations should be borne in mind when deciding the relative size of up-dip and down-dip fields:

1. The depth of vertical shafts increases along with the expansion of the up-dip field.

2. From the viewpoint of the cost of haulage, both variants may be considered approximately equivalent—while the movement of coal down the mine slopes is facilitated by gravity, its subsequent hoisting through the vertical shaft requires a corresponding amount of mechanical power. Besides, in large modern mines, mechanical equipment is used both in transporting the coal up the inclines and in lowering it through the mine slopes, with a substantial part of haulage cost charged against labour, and this is almost equal in both instances.

3. The mining of down-dip fields requires installing supplementary drainage pumps, which are not needed in an up-dip field.

4. In a down-dip field the ventilating current enters by *descending*. Its flow is counteracted by the *natural draught* of the heated air which tends to *ascend*. Ventilation of "downhill" openings presents more difficulties than that of up-dip ones. In mines exposed to firedamp hazards the down-dip field should be supplied with separate air currents in each of its flanks. Therefore, to develop the levels, *three* openings are driven alongside each other—a mine incline and two adjoining manways.

5. There must be at least two passages for men to the shaft level from the down-dip field, but communication with working places in downhill openings is more difficult. In a down-dip field, there should be means for the mechanical transportation of men patterned along those of inclined shafts (Section 6). There can be an exception to this rule only when the vertical distance between the ultimate elevation marks of the mine incline is not in excess of 25 metres.

6. Viewed from a purely economic angle, when the up- and down-dip portions of the mine field are equal in size, the production cost of coal in the first instance is slightly lower. This difference in cost, however, is often disregarded, and in practice, as mining operations progress, down-dip fields become bigger than the up-dip ones. This is done to eliminate the costly and labour-consuming process of deepening vertical shafts in a working mine or of building new mines to exploit deeper lying levels down the dip. However, the economic expediency of long mine inclines is, generally speaking, rather dubious. This applies particularly to "stage" inclines, that is, to inclines equipped with separate hoisting installations capable of transferring the mineral from one installation to another.

One major disadvantage of stage inclines is that they require a large number of workers to service them, and this brings down efficiency per underground worker. For this reason stage inclines should be avoided and long continuous inclines without a transfer of payload arranged instead.

The greater the pitch of the seam, the more the above-cited disadvantages of mine inclines are felt. Therefore, their use is usually limited to an angle of dip of not more than 30-35°.

In order to obviate the necessity of arranging long mine inclines, which are inconvenient and uneconomic, especially in the case of steeply dipping seams, new, lower levels may be developed by deepening vertical shafts and crosscutting. This method is set forth in Section 8.

According to the Safety Rules, when two vertical shafts serve as exits from the underground workings to the surface, they should be provided with ladder ways in addition to mechanical hoisting plants. If the shafts are less than 70 metres deep and both provided with ladder ways, one of them may be without mechanical hoisting apparatus.

## 8. Opening Through Vertical Shafts and Crosscuts

A *slightly dipping* or *tilting* seam (with an angle of dip ranging between 25° and 45°) may also be opened through a vertical shaft with crosscuts (Fig. 17). To avoid driving separate crosscuts to each

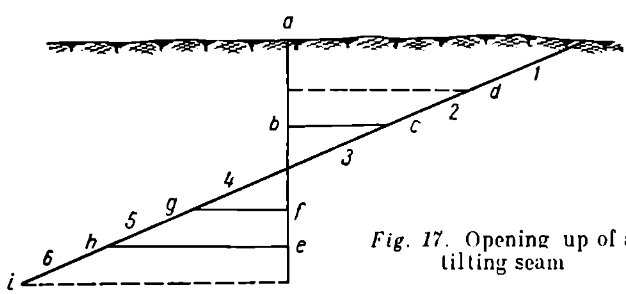



*Fig. 17.* Opening up of a tilting seam

level of the mine field and to reduce the number of shaft stations, one should do as follows. The first, uppermost level is developed without driving any crosscut, and the coal mined is passed along permanent mine slope *cd* down to crosscut *bc*, running from shaft *a* to the haulage entry at the second level. Similarly, to develop levels in the down-dip field, it may prove advantageous to drive crosscuts *fg* and *eh* to open up the fourth and fifth levels, but to avoid running a long crosscut for the development of the 6th level, this may be opened through permanent incline *hi*.

Early development of new levels through permanent mine inclines or through the deepening of shafts and crosscuttings has substantial advantages and drawbacks.

It has already been mentioned that operation of permanent inclines requires special facilities for transporting payloads and men, complicates ventilation and infrequently demands considerable outlays for maintaining inclined mine openings. Most important, however, is the fact that permanent inclines require additional service personnel and this reduces overall efficiency per man at the mine.

These shortcomings may be eliminated if mine levels are developed through crosscuts. This method, however, is extremely complex and costly work, for it requires deepening vertical shafts and excavating and equipping shaft stations and crosscuts.

On the other hand, from the standpoint of operating costs, crosscuts offer greater advantages and conveniences than permanent inclines.

For this reason either of the above-mentioned methods of developing new levels may prove more profitable economically, this depending upon local conditions and, chiefly, on the seam's angle of dip. The proper choice requires a technical and economic comparison of all possible alternatives in accordance with the rules set forth below (Section 18).

*High-dipping* seams with a pitch angle of 45-90° are opened up almost exclusively through vertical shafts and crosscuts. Their early development through inclined shafts is seldom practised, and only in mines with low annual output and shallow shafts.

While in mining gently dipping seams it is possible, if necessary, to increase the inclined height of the level interval to quite sizable proportions, in the case of high-dipping ones it is limited to a narrower range. This is due to difficulties engendered by roof control, men's movement and delivery of timber and other supplies. On the other hand, the greater the level interval, the less the cost charged against one ton of mine output for the excavation of shaft stations and crosscuttings. That encourages increasing the level interval.

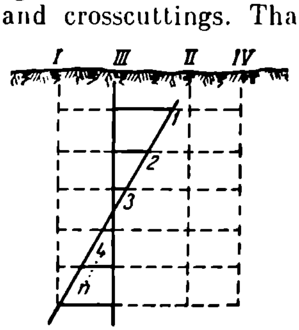



*Fig. 18.* Opening up of a high-dipping seam

On account of all these contradictory factors, the vertical height of a level interval in thin high-dipping seams and in those of medium thickness is usually set at about 100-130 metres, while in thick seams, where the increase in the height of the level interval causes greater difficulties, it generally ranges between 80 and 100 metres.

The opening of steeply dipping seams is almost invariably effected through vertical shafts, from which *level crosscuts* are driven to each level (Fig. 18). With the shaft located at site *III*, that is, intersecting the seam approximately midway down its length, the aggregate extent of crosscuts is minimal compared to locations *I* (with the shaft sunk in the rocks of the hanging wall only) and *II* (when the shaft is driven in the rocks of the foot wall only). But all these three locations have substantial disadvantages of their own. Extraction of the mineral in the seam causes shifts of hanging wall rocks, which may have repercussions on the ground surface too. These shifts may have an adverse effect on the condition of the shafts, crosscuts and shaft or bottom stations excavated in the hanging wall rocks, as well as on the surface structures of the mine. To eliminate these untoward effects, it is necessary to leave an extremely large protective coal pillar under all above-named underground workings and surface plants and structures, and this would reduce the percentage recovery of the mine field reserves. For this reason location *I* of the shaft is regarded unacceptable. As stated before, one advantage of location *III* is the minimal total length of crosscuts, but the considerable waste of coal in protective pillars does not favour adoption of this alternative either. When the shaft is sunk over its entire length in foot wall rocks (*II*), the displacement of hanging wall rocks does not affect the stability of the shaft, bottom stations and crosscuts. The fact should be taken into account, however, that in a steeply dipping seam or vein extraction of the mineral may be followed by a slide of the foot wall rocks too. Therefore, it is better somewhat to increase the aggregate length of crosscuts and drive the shaft at some distance from the outcrop of the deposit at point *IV*. That would obviate the possibility of the underground workings and surface structures of the mine being damaged by sliding foot wall rocks and make it unnecessary to leave any safety pillars.

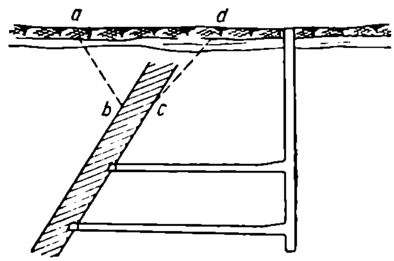


*Fig. 19.* Opening up of a thick high-dipping seam

In opening up high-dipping seams, shafts should be sunk in their foot wall only (Fig. 19), this in consideration of the possible extraction of the upper portion of the deposit through open pit *abcd* and of the layout of surface plants and buildings precluding the necessity of leaving any protective pillar beneath them. Because of the considerable thickness of the seam, the abandonment of this pillar would mean an excessively high loss of coal.

In steeply dipping deposits the levels in a mine field are invariably worked in the descending order. When the levels are mined in upward sequence, it is necessary first to drive the shafts down to their ultimate depth. Besides, there would be excavated areas beneath productive levels, and in a steeply dipping deposit this might cause appreciable difficulties following the shifting of rocks. For this reason the levels in a high-dipping deposit are mined exclusively in descending order, starting with the one nearest to the surface.

Let us follow the sequence of early development operations as depicted in Fig. 20. The main hoisting shaft is first sunk down to the point of the haulage entry of the first level (Fig. 20*a*). To secure ventilation it will suffice to sink the air shaft only to the point marking upper or ventilating entry of the same level. Haulage and ventilating crosscuts are pushed forward from the shafts to permit excavating

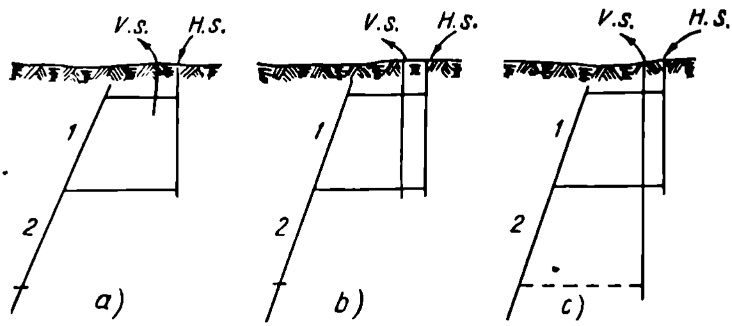



*Fig. 20.* Sequence of level development in a steeply dipping bed

main level entries, that is, to start development and then stoping work in the first level. The direction of ventilating air currents will then be as follows.

The intake current of fresh air will enter the mine through the hoisting shaft and pass to the haulage entries in both wings of the mine field via the lower crosscut of the first level. Ascending, it will circulate along stope faces, flow to the ventilation crosscut and will then return to the surface through the ventilating shaft. This movement of the air is kept up by the operation of an exhaust fan set up over the mouth of the upcast shaft. In Section 6 of this chapter it was pointed out that the driving of the ventilating shaft and crosscut is sometimes dispensed with altogether in mining the first level, and that the return airways are connected every few hundred metres with the ground surface through pits serving as communication openings for the passage of men, lowering of supplies and ventilation. In this case ventilation is effected either by exhaust fans set up over every consecutive pit or else by making the main hoisting shaft serve as the passage for the downcast air current fed by a pressure fan.

While it is enough to sink the air shaft down to the upper entry of the given level to assure adequate ventilation, to facilitate communication between the haulage level and the ground surface and to make better use of the ventilating shaft as an auxiliary opening its depth should preferably be equal to that of the main hoisting shaft (Fig. 20*b*).

For purposes of raising the hoisting shaft, it is desirable to drive the ventilating shaft in advance by one level (Fig. 20*c*). When mining high-dipping seams, the periodic deepening of shafts for the development of new levels hampers the routine operation of shaft stations in one way or another. On the other hand, preliminary sinking of the ventilating shaft to a new level and subsequent raising of the hoisting shaft reduce to the minimum the time needed for switching over the operation of the main hoisting plant to a new level.

Efforts required for opening and developing new levels in steeply dipping deposits consume a great deal of labour and time. By analogy to what has been said above with reference to the development of new levels in opening up a mine field through inclined shafts (Section 6) this work should be started well ahead of time.

## 9. Opening of a Horizontal Bed

Entries or drifts made along the strike of a bed occurring at a certain dip run horisontally. In a flat dipping deposit, where the notion of strike loses its significance and mine workings driven in the plane of the bed in all directions are level, this factor falls away altogether. Therefore, when flat dipping beds or seams are worked, the mine field sometimes is not subdivided into sections analogous in outline to levels, but are cut into *panels* which may be oriented to each other at different *angles* depending upon the features specific to the run of the bed bottom, or have nonrectangular contours (Fig. 21).

Nature knows no beds whose occurrence is geometrically horizontal. Each has an "undulating" bottom with local "sink holes" and "upheavings". This irregularity of occurrence is of great importance for the choice of workings to be provided with mine tracks and drain ditches.

Hence the different orientation of panels, in accordance with the relief (*hypsometry*) of the bottom.

In the U.S S.R. flat occurrences of variable thickness are common to many bedlike deposits of the Moscow coal basin. Coal measures there usually have but one payable seam. Coal occurs in the shape of lens-like bodies (deposits) of irregular outline, lying close to the surface, usually a few scores of metres. The lenses are usually about 1-2 km long and 0.6-1.5 km wide, although some deposits cover larger areas with continuously occurring coal seams. More often than not, seams or beds are 1-3 metres thick, but quite frequently their thickness tends to vary over short distances, sometimes reaching 4-6 metres.

In the outline and nature of their occurrence, the mine field areas of coal deposits in the Moscow basin are usually irregular. Ordinarily they are opened by centrally located twin shafts which, because of the "undulating" bed bottom, are whenever possible sunk in low-lying ground to ensure an adequate runoff of mine water to the main water collectors or sumps in the vicinity of the shaft station. Quite often, however, auxiliary pump stations have to be set up. Because of the shallowness of coal beds, proper servicing of individual sections of the mine field not infrequently requires the arrangement of air pits with ladder ways for the passage of men, in addition to the main shafts.

Because of the irregular contours of the mine fields and nonuniform run of the bed bottom the main entries are driven in different directions, breaking the mine field into separate portions which are mined individually by systems described below (Chapter XIII, Section 5), or else are preliminarily divided into panels.

Flat bedding of coal seams is also common to Cheremkhovo coal fields and some other areas. Combustible shale deposits in the Baltic area and the Middle Volga are characterised by their nearly flat pitch.

Nearly all the coal deposits in the U.S.A. are practically flat. Quite apart in this respect is the Pennsylvanian anthracite district, whose geological structure roughly resembles that of the Prokopyevsk-Kiselyovsk district of the Kuznetsk coal fields, though the properties of their coals are different.

## 10. Division of a Mine Field into Panels

In the foregoing text it was assumed that in the instance of nonhorizontal seams the mine field had to be divided into levels permitting proper sequence of mining.

But, as already mentioned, there also exists a method of dividing a mine field into *panels* applicable to the development of seams with a certain angle of pitch.

One of the variants of this method provides for the following operations (Fig. 22). The mine field is divided into an up-dip and down-dip portions, each cut into panels *1*, *2*, *3*.... The entry driven on the level of the shaft station, that is, the one delimiting the up-dip and down-dip fields, is called *main entry*. Up and down the dip each panel is bounded by the main entry and the mine field boundary, while in the direction of the strike it borders on the adjacent panels or on the nearby panel and the mine field boundary. Each panel in the up-dip field is provided with an independent permanent *panel slope*, and in the down-dip field with an individual *panel incline*.

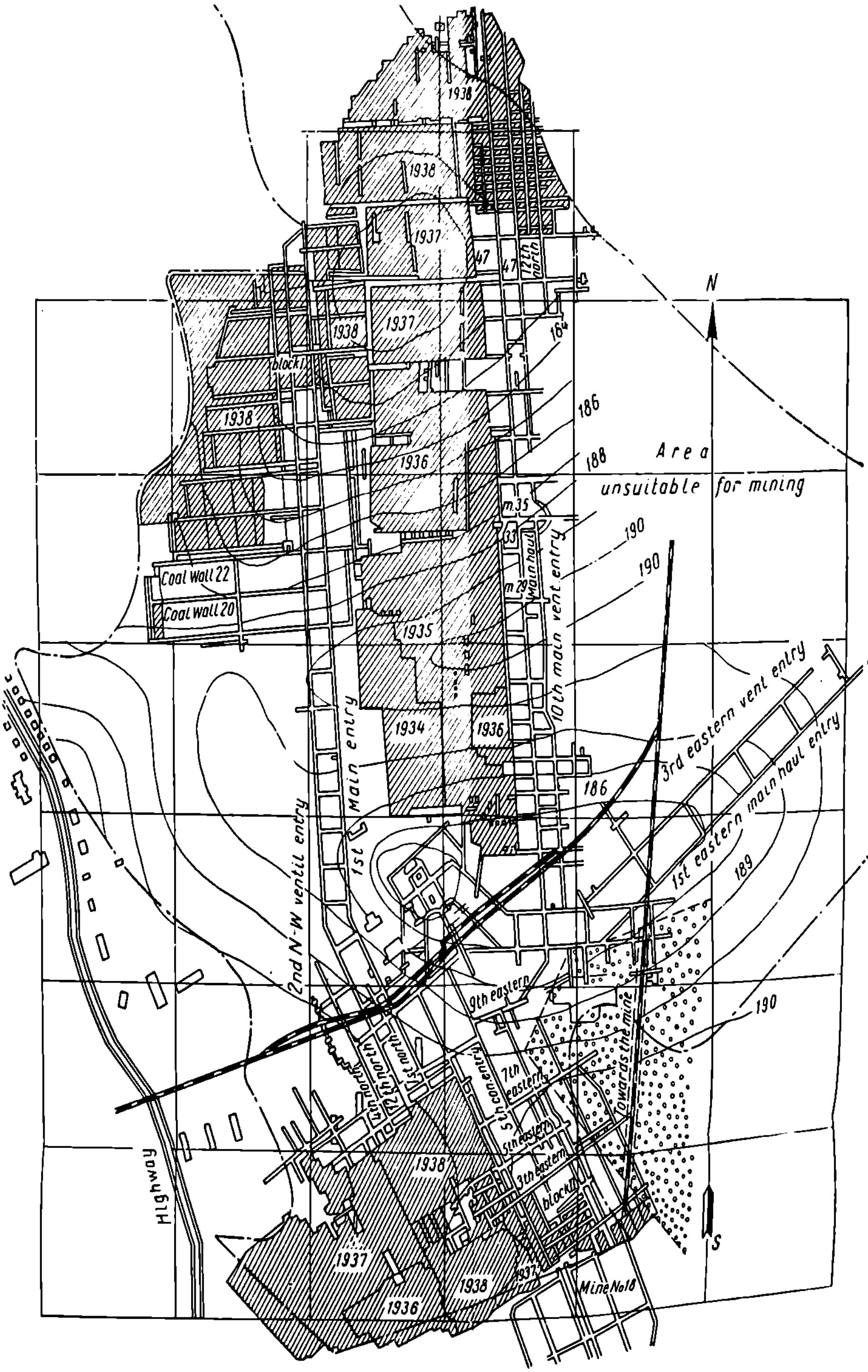



Fig. 21. Opening up of a mine field in the Moscow coal basin

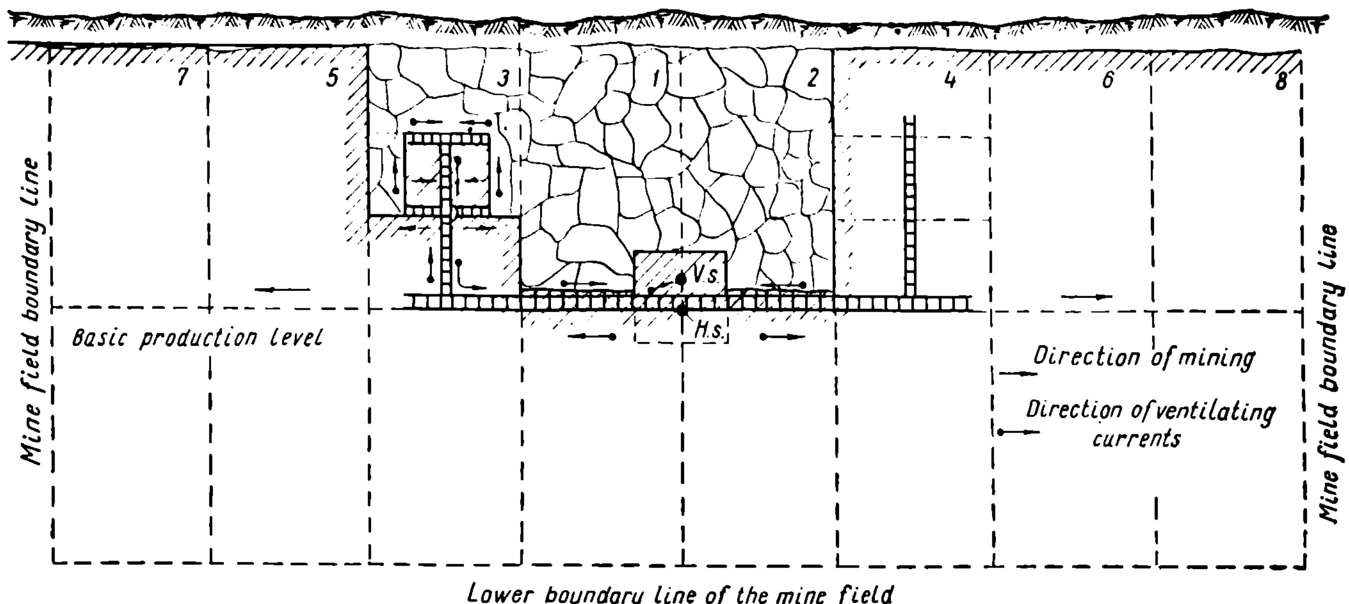



*Fig. 22.* Division of a mine field into panels. Variant I

To increase the breast front of stoping each panel, as a rule, is worked bilaterally. The width of the panel (that is, its extent along the strike) is set at several hundred metres. Thus, the concept *panel* may be defined in the following manner: it is the part of a mine field bounded by the main haulage entry of the basic production level and serviced by the independent tramming face entry directly abutting on this main entry. A schematic position of the stoping breast front is shown in Fig. 22. The portion of the panel worked at the same time is called *stage*. In Fig. 23 each panel has three stages.

Panels may be worked in sequence *1*, *2*, *3*, *4*... alternately now in one and now in another flank (assuming that the breast front of stopes in one panel is sufficient to secure planned output from the seam in production). This method allows extraction of the mineral from panels nearest to the shaft. On the other hand, however, it makes it imperative simultaneously to drive and maintain main entries in both wings of the mine field. Panels may also be mined in sequence *1*, *3*, *5*, *7*, *2*....

One major disadvantage of this variant of the panel method illustrated in Fig. 22 is the direction of the main ventilating currents: the intake current of fresh air and the return current move side by side along the openings of the main production level. This leads to leakages and short circuits of air currents, and such panel development in mines with high firedamp evolution should, therefore, be avoided. To do away with this very serious handicap, it is suggested that the main airway be driven as a lateral entry, the more so since its service life is long.

This disadvantage of the panel development of the mine field may also be eliminated by using the alternative shown in Fig. 23, that is, by making return air currents move towards the ventilating entry driven near the upper boundary of the mine field. A very substantial disadvantage of this method, however, is that extremely long slopes (or inclines in the down-dip portion of the mine field) have to be maintained in mined-out areas. Generally speaking, the negative feature of the panel layout is the necessity inherent in this method of driving a series of long oblique openings—permanent mine slopes and inclines.

As the upper boundary of the mine field does not lie too deep from the ground surface, each panel can be ventilated through an independent upcast pit, and in this case no common upper entry is needed.

One advantage of the panel layout is the mine field's single main level with electric haulage, while panel entries are equipped with conveyers only.

Another substantial advantage of the panel layout is that, when needed, it is possible to develop a large footage of working stopes in one seam.

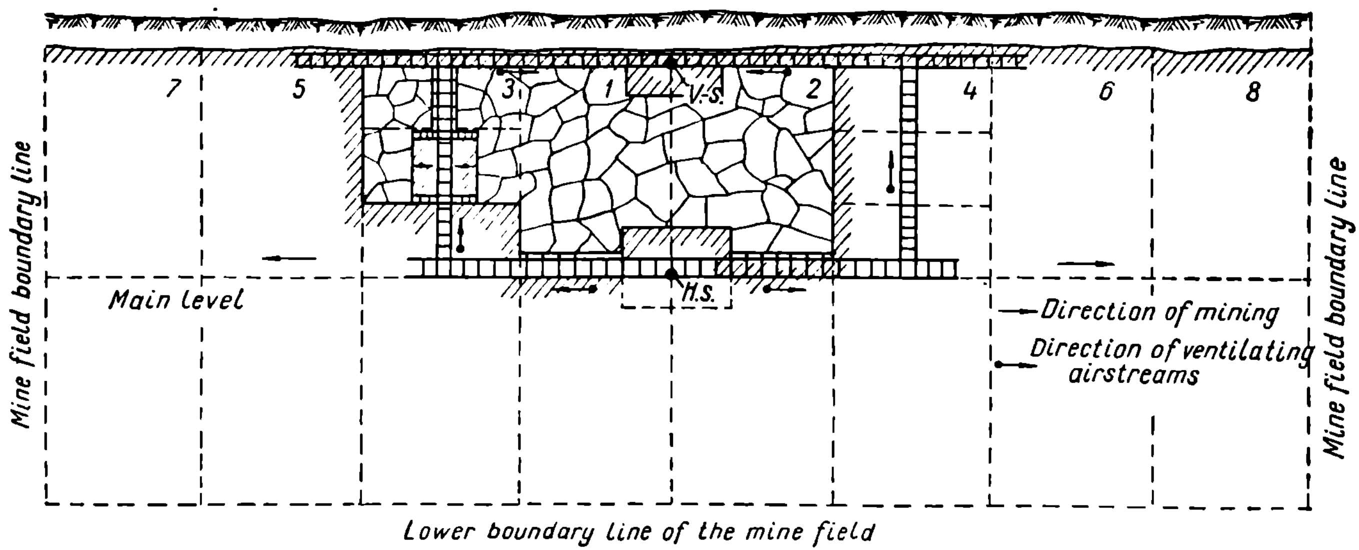



*Fig. 23.* Division of a mine field into panels. Variant II

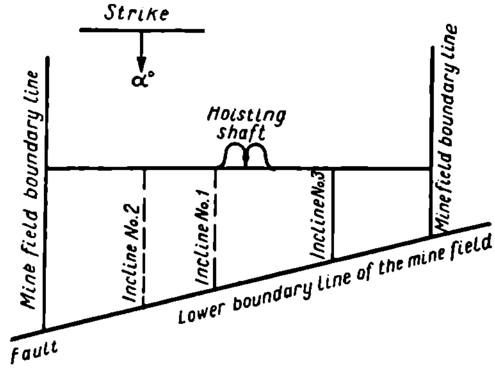



*Fig. 24.* Opening up of a mine field down the dip through several permanent inclines

It was Prof. B. Bokiy who first compared the opening and development of a mine field by using levels and panels (in a work named *Mining by Fields of Big Height*). His conclusions favoured the level method of mining. At present, however, this problem should be investigated again since, though it has not yet been proved by calculation, the new mechanised transport facilities may broaden the scope of the panel development application.

To form a proper judgement on comparative advantages and disadvantages of the level and panel methods of opening and developing a mine field it is essential that this comparison be complex, that is, it should include both the technical and economical aspects of the driving of mine workings, their maintenance, transportation of coal, barren rocks and supplies, passage of men, ventilation, power supply, and for down-dip fields—mine water disposal as well. The answer to this question to a first approximation may be found through estimating the number of men required and their work in both these methods.

It should be noted that the level method of development is simpler and may be employed within a wider range of geological conditions.

There have been instances of down-dip fields being worked through individual inclines. This occurs when it is impossible to drive the lower entry across the entire mine field on account of the oblique outline of the lower mine field boundary caused by the geological disturbances of bedding (Fig. 24).

## 11. Opening of Coal Measures

In most cases, coal seams in deposits do not occur singly, but in *measures*. If seams $p_1$, $p_2$, $p_3$ ... (Fig. 25) in the measure lie far from each other, it may be technically advisable and economical (this being decided by calculation in planning the mining of a particular deposit) to open them through *separate* shafts Nos. *1*, *2*, *3*....

When the seams of the measure occur closer to each other, they can be developed and worked *jointly* through one shaft. In combined development, the seams are interconnected by mine workings, which, depending largely upon the angle of pitch of the seams, may be located in many different ways (Fig. 26).

When two seams $p_1$ and $p_2$ (Fig. 26*a*) are flat or nearly so, the length of the vertical connecting opening is minimal. This may be a blind shaft intended for hoisting (continuous arrow in the drawing) or lowering loads (dash arrow). To simplify transportation and facilitate the movement of men, the connecting opening (dash line in

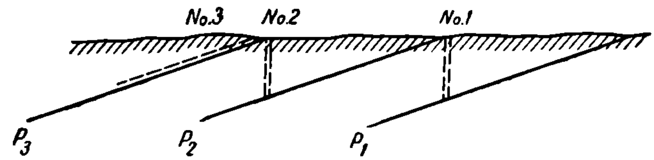



*Fig. 25.* Opening up of coal-measure strata through separate shafts

Fig. 26*a*) may be driven as an incline in the country rock (for hoisting loads), or as a slope or winze in the country rock (for lowering loads).

When the seams occur vertically (Fig. 26*b*) they are connected by a crosscut.

Seams with an angle of dip $\alpha$ other than 0° and 90° (Fig. 26*c*) may be connected by horizontal crosscuts, vertical blind shafts, slopes or, finally, by inclines and slopes excavated in the country rock. The smaller the pitch of the seams, the greater the relative length of the crosscut and the more it costs to drive a crosscut than to excavate shorter vertical or inclined openings. But being horizontal workings, crosscuts, in terms of operational convenience, possess such vast advantages that *in combined mining it is generally preferrable to use them to connect individual seams.* Although the excavation of crosscuts requires greater outlays, they help to reduce operational expenses. Transportation—electric haulage, for example—in entries and crosscuts proceeds in this instance uninterruptedly, whereas connection of seams by a vertical or inclined opening disrupts the continuity of this process and requires additional transport facilities in these workings, and that entails an increase in operational costs.

Bearing all that in mind, let us examine the characteristic layouts used in the development of coal-measure strata.

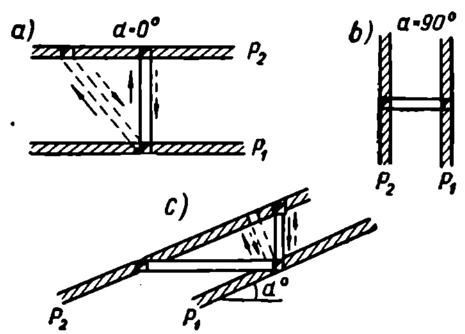



*Fig. 26.* Possible methods of connecting seams by mine openings

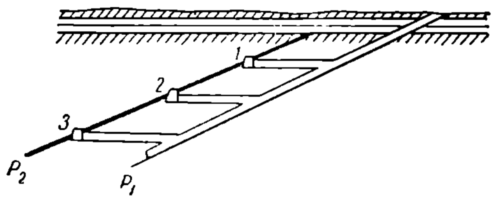



*Fig. 27.* Combined opening up of seams through inclined shafts

In conditions favouring the sinking of inclined shafts (Section 6), two or several seams $p_1, p_2$ ... may be developed by this method (Fig. 27). To reduce coal wastage in protective pillars, the inclined shafts are driven in the lowest seam of the measure. The seams are connected with each other by level crosscuts. Underground workings may be aerated with the aid of the draught produced by one main or two individual fans.

Opening of two flat or gently pitching beds located at a considerable distance from each other through one common main vertical shaft (Fig. 28) may involve driving separate bottom stations *a* and *b* at the intersection of these beds by the shaft, with the further opening, developing and stoping of the beds carried out quite independently. In this case hoisting plant in vertical shafts should be arranged so as to allow simultaneous operation on two levels. However, since it is extremely inconvenient (though not impossible) permanently to operate the hoisting plant now on one and now on another level, it is preferable to use two independent hoisting plants in such conditions.

When the distance between flat beds $p_1$ and $p_2$ is smaller, they may be connected by a vertical blind shaft. The hoisting then is done from one level only. If the coal extracted from bed $p_1$ is *hoisted* up the blind shaft (continuous arrow in Fig. 29), the coal coming from both beds is brought up to the ground surface from shaft station *a* by the main hoisting plant. This obviates the necessity of sinking the main shaft down to bed $p_1$. But if coal from bed $p_2$ is passed down, the coal mined in both beds is to be hoisted mainly from shaft station *b*, driven in bed $p_1$. The vertical opening connecting these beds

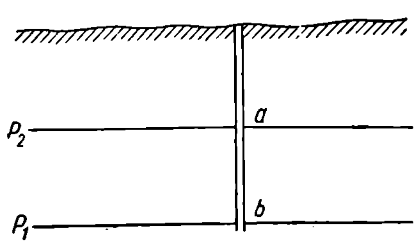



*Fig. 28.* Opening up of two flat seams through a vertical shaft

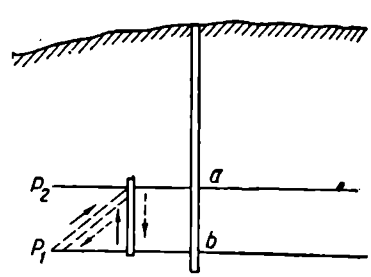



*Fig. 29.* Opening up of two flat seams through a vertical main shaft and a blind shaft or winze

may be replaced by an inclined one (dash line in Fig. 29), if this is deemed advisable on technical and economical grounds which is determined by preliminary estimates.

To secure adequate ventilation for underground workings and to provide for no less than two escape openings to the surface, supplementary mine workings are excavated (they are not shown in Figs. 28 and 29).

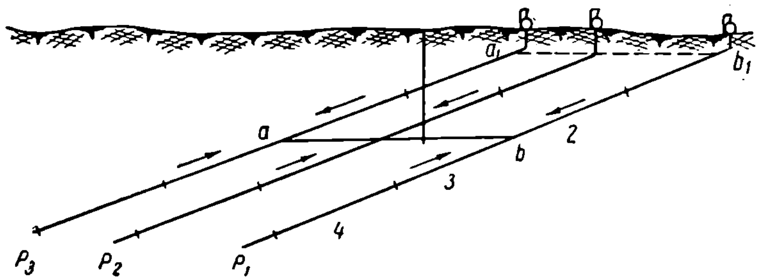



*Fig. 30.* Opening up of coal measures through a vertical shaft and a permanent crosscut

A widely practised method for opening up coal measures is illustrated in Fig. 30. The strata of gently pitching seams $p_1$, $p_2$, $p_3$ ... are opened by a vertical main shaft, from which permanent crosscut *ab* is driven. The levels (or panels) within the range of each seam are opened and developed in the fashion described in Sections 7 and 10, that is, as though the mine field of any particular seam were opened through a vertical shaft running along the line of intersection of the seam and the permanent crosscut. The coal broken in the up-dip field is passed down the mine slopes to the level of the crosscut, while that coming from the down-dip field is brought up along the permanent mine inclines. Ventilation is effected by individual fans set up for each seam, or else ventilation crosscut $a_1b_1$ is driven in the upper portion of the mine field to collect return air currents which are then discharged to the ground surface with the aid of a single exhaust fan.

To avoid excavating long mine slopes and inclines, one may drive, as illustrated in Fig. 17, not one, but two and even more crosscuts *I*, *II*, *III* (Fig. 31) in combination with permanent mine slopes (in Fig. 31, stoping of the first level) and inclines (stoping of the fifth level).

When space between individual seams in the measure differs widely, the above-mentioned methods of mine opening may be employed *in various combinations*. For example, Fig. 32 is illustrative of an instance when one of the seams $p_3$ lies at a very considerable distance from two other seams $p_1$ and $p_2$, which are contiguous. It may prove economical to approach the outlying seam through a

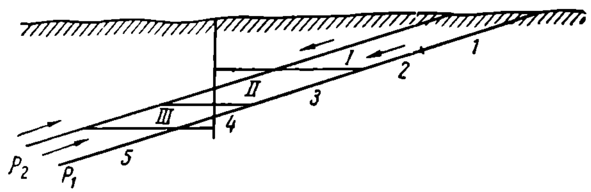



*Fig. 31.* Opening up of a seam series through a vertical shaft and crosscuts

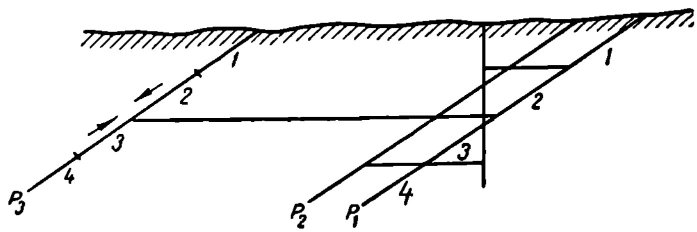



*Fig. 32.* Combined method of developing a series of coal seams

long crosscut and to develop the levels in this seam through a permanent mine slope or incline.

Fig. 33 depicts a typical layout for opening up a steeply dipping coal measure. Here continuous lines show the sections of shafts and crosscuts which must be excavated within the area of the first level to secure the progress of stoping (it is assumed that the hoisting and air shafts are sunk to the same depth). The position of shafts and crosscuts for the development of subsequent levels is shown by dash lines. The circulation of ventilation currents is marked by arrows. If only the first level is aerated, no ventilation crosscut is needed when pressure fans are used. In this case, the return air from the work-

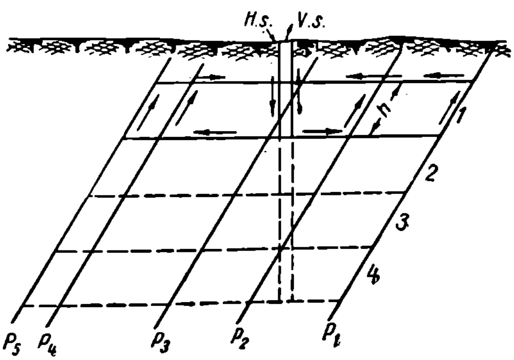



*Fig. 33.* Opening up of a steeply dipping coal measure

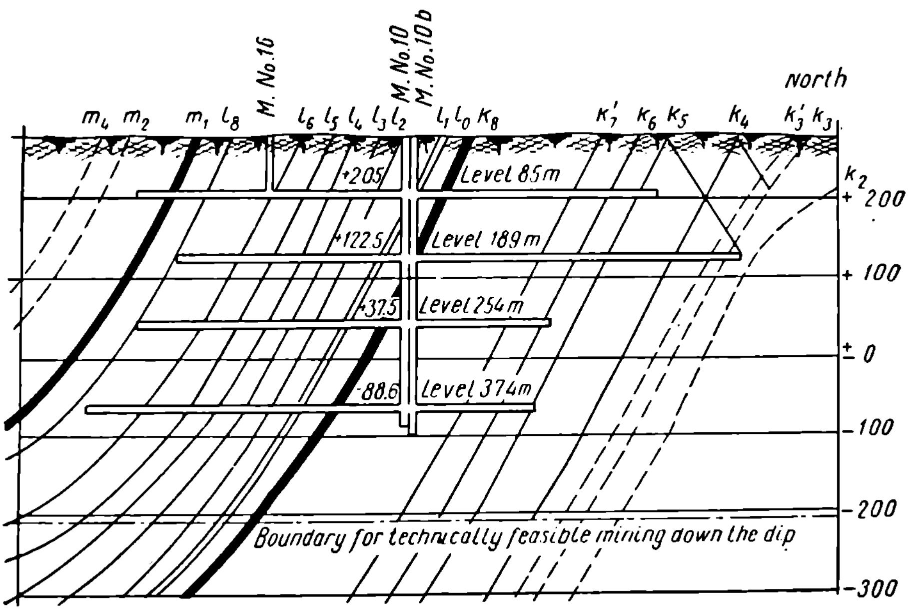



*Fig. 34.* Opening up of a series of thin, high-dipping seams in the Donets coal fields

ings of each seam is discharged to the surface via auxiliary pits, which at the same time serve for lowering men, timber and other supplies into the mine. When there is a considerable number of productive seams and rugged surface topography, this method is inconvenient, since it necessitates sinking many auxiliary pits, through which it may prove difficult to deliver supplies. That is why ventilation crosscuts of the first level are considered indispensable at bigger mines.

In the mining of each subsequent level, every former haulage crosscut of the preceding level serves as an airway.

Fig. 34 illustrates a typical layout for opening a series comprising numerous thin, high-dipping coal seams occurring in conditions specific to the main anticline of the Donets coal fields.

If all the seams of the series were mined simultaneously and over equal areas then, following the procedure similar to that set forth in Section 4, we would arrive at the following relation between annual mine production $A$, on the one hand, and level interval $h$, average annual advance of stoping operations in one wing $L$, total coal output per 1 square metre of seam area $\Sigma p$ and the mean coefficient of coal recovery $c$, on the other:

$$A = 2Lh\Sigma pc.$$

As before, this formula is set forth on the assumption that only one productive level is mined at a time and that the mine field has two equally large flanks. But in developing coal measures it is not possible to work all the seams simultaneously and uniformly. This is primarily due to the fact that, as a rule, the seams in a series must be worked in definite order. After the extraction of coal from seams there occur rock displacements over the mined-out areas, in the form of collapses, fissures, subsidence and sagging. Therefore, if there is an unstoped seam lying over the one mined at any given moment, the former may be "undermined" from below during extraction of the latter. This would mean disturbance of the continuity of coal and enclosing rocks of the upper seam, caused by rock shifting due to the extraction of the lower seam and that would complicate further mining operations. The degree of damage caused by "undermining" may be of widely differing proportions, ranging from complete impossibility of proceeding with the mining of the upper, undermined seam to almost no injurious effect at all, this depending on existing conditions (thickness of the seams, distance separating them, angle of dip and properties of rocks occurring between the seams).

When the damaging effect of undermining from below has to be reckoned with, the principal measure that can forestall this dangerous condition is mining of upper seams *before* the lower ones. In other words, if the seam worked at any given moment has another productive seam over it, and the conditions favour its undermining from below, *the upper seam should be mined first.*

Thus, to eliminate undermining from below, the upper seams should be worked before the lower ones; in other words, all the seams in the mine field cannot be mined simultaneously and uniformly. Hence, concurrently mined are seams whose aggregate output is less than the value $\Sigma p$, and this is set at $k\Sigma p$, where $k$ is the *variation factor* of coal extraction from the seams, that is, the number showing what portion of the aggregate output capacity of the seams in a series is utilised on the average at a time. In planning mining operations the actual value of the variation factor must be determined after a detailed consideration of the sequence in which seams are to be mined in the series and estimation of the rate of advance or the lead to be adopted in working the overlying seams.

If a deposit is very rich, it may happen that, in order to ensure an adequate stope footage in an active mine, or the planned tonnage for a newly projected one, it will suffice to work only a portion of the seams in the mine field at a time. This is the second reason prompting the inclusion of the variation factor in all the estimates of this kind.

Thus, in mining coal measures, the formula above should read

$$A = 2Lhk\Sigma pc, \tag{3}$$

where $k$ is the variation factor.

Correspondingly, the formula for estimating the level interval will be

$$h=\frac{A}{2Lk\Sigma pc}\,. \tag{4}$$

Explanations made in Section 4 about the values included in these formulas as applied to the mining of one seam are also relevant to working coal measures.

## 12. Opening of Disturbed Deposits

It was assumed above that beds in a mine field occurred more or less regularly.

When the deposit is disturbed by folds, faults and other dislocations, the methods applied for its opening must be adjusted to each individual case and in this instance it is difficult to set forth any general guiding principles or rules. More often than not, however, badly disturbed or disrupted coal deposits are opened through vertical shafts and level crosscuts.

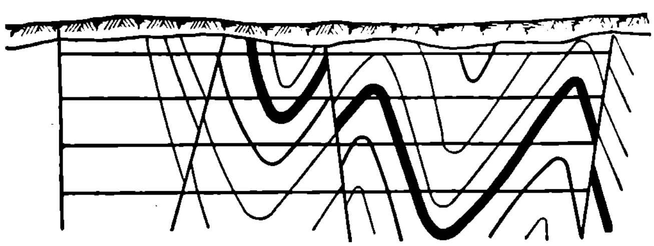


*Fig. 35.* Opening up of faulted deposits in the Prokopyevsk-Kiselyovsk district of the Kuznetsk coal fields

One characteristic example is the development pattern of the coal deposits occurring in the Prokopyevsk-Kiselyovsk district of the Kuznetsk basin, where sheets of coal of varying thickness occur in the shape of sharply pronounced folds with steep sides or limbs and are, moreover, dislocated by faults (Fig. 35).

Another example of opening is shown in Fig. 36, where the series of gently pitching seams with a disrupted mode of occurrence is intersected by level crosscuts—haulage *ab* and ventilation *cd*—pushed forward from vertical shafts. Some sections of the seams lying between geological faults may be worked out, depending on their position in relation to crosscuts, in a variety of ways—by mine slopes *1*, mine inclines *2* and blind shafts *3* and *4*.

The layout of haulage crosscut *ab* shown in Fig. 36, however, has a big drawback: with the progress of mining operations it becomes surrounded by mined-out areas. This entails leaving large protective pillars and consequently wastage of coal. Therefore, the following method for opening up a series of gently pitching, disturbed coal beds may be employed at big mines: the haulage crosscut is driven in the foot wall of the measures (dash line *a'b'* in Fig. 36), while from this *lateral entries* are pushed forward and connected with individual sections of the seams by means of blind shafts of different extension, the latter depending on the relative disposition of the seams.

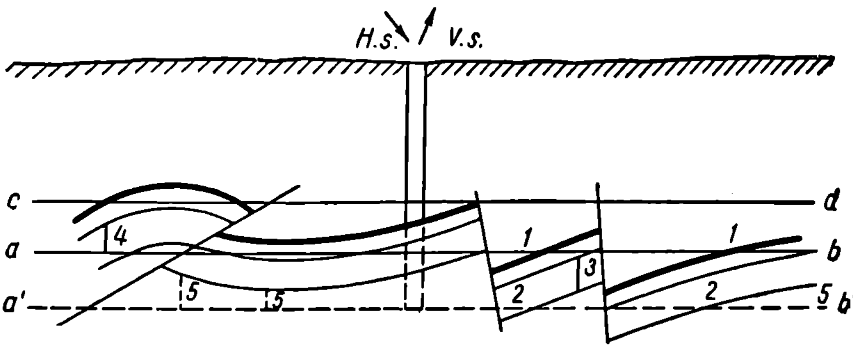



*Fig. 36.* An example showing the opening up of a badly disturbed deposit with a predominantly gentle dip

With this alternative mode of development, the haulage crosscut and entries as well as blind shafts *5* are not exposed to hazards arising from the displacement of mine rocks. Another advantage of this method is the possibility of driving lateral entries in country rocks in a straight line, quite irrespective of geological disturbances in the occurrence of the beds. However, this mode of opening is rather costly and is profitable only in mining measures containing large quantities of coal.

In preparing detailed plans for opening and developing individual sections of faulted deposits, the task of the engineer charged with planning is to find a way of working them by the simplest and cheapest method, by exercising sound judgement and taking into account the peculiarities distinguishing the occurrence of beds within the range of each section of the mine field. *At the same time care should be taken that the aggregate of such sections mined in the course of the year should ensure the planned annual production capacity of the mine.*

## 13. Advance and Retreat Working of a Mine Field

As stated above, individual levels and the mine field as a whole may be worked by *advance mining*, that is, in the direction from the shafts to the boundaries of the mine field, or *by retreat mining*—from the field boundaries towards the shafts. Let us discuss the advantages and shortcomings, as well as the scope of each of these two methods, first in mining individual seams and then their series.

To facilitate comparison, advance mining in working a level is shown in Fig. 37 on the left side and retreat mining on the right. In this instance, the ventilating shaft is located in the centre of the mine field.

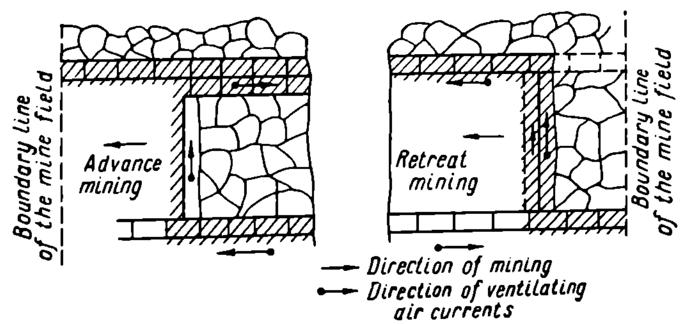



*Fig. 37.* Mining of a level by the advance and retreat methods with a retrograde ventilation scheme

The principal advantage of advance mining is that it allows stoping operations without preliminary driving of level entries over the entire mine field, as is the case with retreat mining.

But there are also substantial disadvantages in advance mining. With this method the haulage entry runs immediately beneath the mined-out area. To protect the strike entry from the direct impact of subsiding rocks, *coal pillars* must be left alongside it. If the size of these pillars up the dip is large, for instance 20-30 metres, the vicinity of the worked-out areas does not much affect the stability of the entry timbering, though it tends to complicate blocking out, that is, driving entries in coal delimiting these pillars, increases the amount of coal lost in pillars and impedes communications between the strike entries and working faces. Should, on the other hand, the sill pillars adopted be smaller in size, say 5-10 metres, this results in high rock pressure on the entry timbering, particularly along the sections closest to the working faces, and necessitates frequent repairs of the timbering and mine tracks. In such circumstances, the ventilating entry, which is flanked on both sides by worked-out areas, is especially vulnerable. If the dip is high, the maintenance of ventilating entry driven in the minable seam proper, except in working very thin seams, is altogether impossible. This is due to the fact that coal pillars, which may be left under the ventilating entry to safeguard it, are actually liable, in conditions prevailing in heavy pitched deposits, to become fractured and gradually sheet and slide down. Another factor contributing to the instability of these pillars is dislocation of rocks forming hanging and even foot walls of the seam mined. All this makes the maintenance of the ventilating entry over the mined-out area impossible. To this one might add that when the entry is supported by rock ribs instead of coal pillars and the rocks enclosing the coal seam are not steady and its pitch is steep, the rock ribs may also slide and that can be attended not only by the development of extremely high rock pressure, but also by a sudden breakdown of the entry. Such cases necessitate the use of a strike entry made in another seam of the series (see below) or of a lateral one driven in country rocks. Cap pillars over the entry are left temporarily, but their subsequent extraction on the ventilating level entails coal losses. And the robbing of floor pillars beneath the ventilating entry in seams with moderate and high dips is not practised at all.

The current of fresh air reaches working faces via the lower haulage entry of the level concerned. Since there is a difference of air pressure between the haulage and ventilating entries, air may leak through holes and cracks in the stoppings arranged in "cross entries" between the pillars, through fissures and crevices in the crushed coal of these pillars and through the mined areas. Such *leakage of air* worsens ventilation conditions at the stopes. Air leakage through fissures in the pillars may cause oxidation, heating and spontaneous combustion of coal, that is, lead to an underground fire. It is by far not always possible to extinguish an underground fire at its source by removing burning and heated coal or with water and fighting underground fires generally boils down to erecting airtight seals, sometimes followed by hydraulic *silting* of the sealed-off section, that is, by filling its workings with *pulp*—a mixture of water and clay particles. The setting of air seals in the main entries makes access to stopes in advance mining of the level impossible and production in this flank of the field has to be stopped altogether until the fire is brought completely under control. On the other hand, it is difficult to isolate a fire by arranging airtight seals in mine workings of secondary importance because it requires a large number of seals. The advance method of mining thus creates very unfavourable conditions for fighting mine fires caused by spontaneous combustion of coal.

The above-cited disadvantages—difficulty of maintaining entries, air leakage and possible outbreak of underground fires—are to a great extent obviated in retreat mining of levels.

The section of the haulage entry which has to be maintained to ensure proper operation of the mine transport is flanked on both sides by coal "in situ' (right side in Fig. 37). Therefore, it is not exposed to massive rock pressure observed in the advance method of mining.

The position of the ventilating entry is somewhat inferior, but from the down-dip side it too is adjoined by an intact solid block of coal. Hence, in retreat mining it can be maintained during the extraction of both the thicker and heavy-pitching seams. The aggregate area of entry chain pillars in retreat mining is thus much smaller and their service life shorter, inasmuch as no pillars at all need be left under the ventilating entry.

As air leakages via worked-out areas are practically ruled out in retreat mining they do not occur at all. This and also reduced coal losses in pillars minimise fire hazards due to spontaneous combustion of coal. Even if a fire should break out, it is simpler and easier to bring it under control. To isolate the focus of the fire, it suffices to set up seals in the main strike entries, to prepare working faces and resume stoping operations, isolating the fire-stricken section from the rest of the area by a pillar until the fire is brought under complete control.

It is to be noted, however, that when the mining method used requires a large number of development workings, there should also be seals arranged in some of these workings so as to reduce the amount of coal left inside the isolated fire-stricken area.

The availability of main level entries driven in retreat mining along the entire length of one of the flanks of the mine field prior to opening production stopes is one of the main advantages of this method. Driving level entries beforehand makes it possible to explore additionally and in detail the features specific to the occurrence of the seam and accurately establish the relative positions of minor geological disturbances, which is of paramount importance for planning stoping operations.

In retreat mining the stopes are always adjoined by a ready tramming entry, this ensuring adequate performance of switching operations in electric haulage. True, this can also be achieved in advance mining, providing the breast of the haulage entry is driven no less than 70-100 metres ahead of the stope face. In practice, unfortunately, this condition is not always abided by and that complicates transport operations.

The above-cited employment of switching operations in electric haulage raises the efficiency of the mine transportation system and, what is particularly important, speeds up the handling and removal of broken coal from the stope areas.

The working of levels by the retreat method has its shortcomings too.

To start sloping in a level by the retreat method, it is necessary first to push forward a haulage entry over the entire length of the mine field and simultaneously drive and maintain an upper (ventilating) entry of the same length. Prior drivage of these openings requires much time and involves certain technical difficulties, largely in connection with the ventilation of their advance headings as well as their supply with electric power and compressed air. To facilitate air circulation, haulage and ventilating entries may, when being driven, be connected by inclined openings, which later may be used as chutes, mine slopes or manways.

The above applies to the ventilation of the level along the central (retrograde) aeration pattern (Fig. 37).

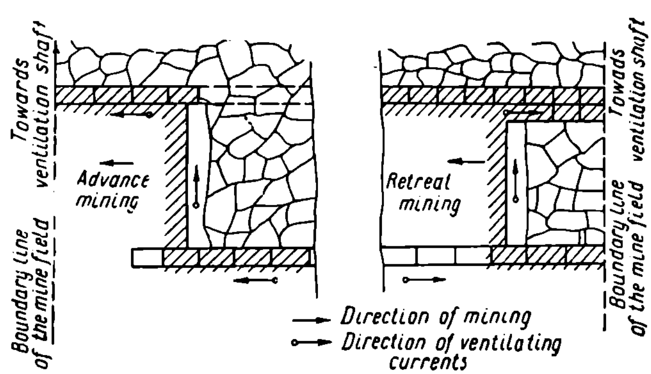



*Fig. 38.* Mining of a level by the advance and retreat methods with a boundary ventilation scheme

The sinking of discharge air shafts on the flanks of the mine field (Fig. 38) nullifies many advantages of retreat mining. Comparison of the left (advance mining) and right (retreat mining) sides in Fig. 38 reveals that in the case of boundary (diagonal) ventilation, the advantages of retreat mining described above apply to the haulage entry. But the ventilating entry would remain in the same unfavourable position as it is in the case of advance mining using a retrograde system of ventilation.

If it proves difficult or even impossible to maintain a ventilating entry driven in the minable seam, it may, as said before, be replaced by a *lateral* entry. To this end, lateral entries $a_1$ and $b_1$ may be made in the country rocks of the foot wall parallel to the main entries $a$ and $b$ (Fig. 39). The entries driven in the seam and country rocks are connected with each other (ordinarily every few hundred metres) by *intermediate crosscuts* $aa_1$ and $bb_1$. The lateral entries are largely used for handling coal, for the passage of men to and from the stopes and for ventilating air currents, while the entries made in the seam proper need not be maintained at all. Hence, the use of lateral entries makes it fully possible to mine the level by the advance method.

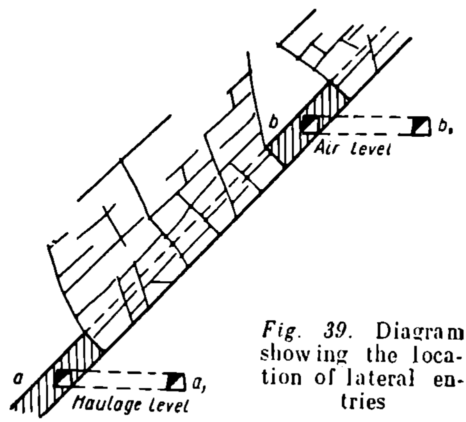



*Fig. 39.* Diagram showing the location of lateral entries

When coal measures are mined (Fig. 40), damage to ventilating entries may be caused not only by the dislocation of rocks following the extraction of the given seam, but also by dislocations occurring in the ones below it. Therefore, the main entries in the series of closely superimposed seams $p_1$, $p_2$, $p_3$ may be connected with each other by intermediate crosscuts with a view to concentrating the passage of men, haulage and the flow of the main air currents in entries *a*

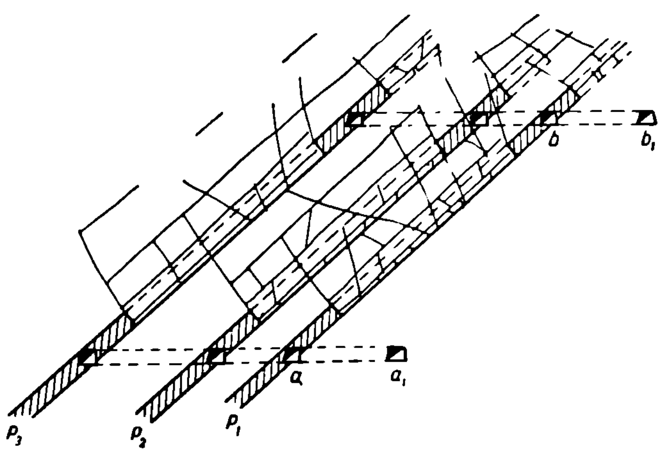



*Fig. 40.* Disposition of group and lateral (fringe) entries

and *b* of the lower seam of the series. Such an entry is, therefore, called *group* or *gathering* entry. Should it prove difficult to maintain the entries made in the lower seam (especially the ventilating one), the group entry may be driven in the country rock too ($a_1$, $b_1$). In all these instances the level may also be worked out by the advance method. The question of combined development and stoping in a series of seams is discussed in detail in Chapter XXII.

Let us sum up all we have said before about the relative advantages and drawbacks of advance and retreat mining of levels.

*Advance mining* allows a quick progress of stoping operations, but it has its shortcomings: 1) difficulties in maintaining level entries, which in the case of a ventilating entry driven in the working seam of medium thickness and heavy pitch may even prove quite impossible; 2) higher coal losses in pillars; 3) air leakages; 4) fire hazards due to spontaneous combustion of coal and difficulty of controlling such fires. The aforementioned disadvantages incident to the mining of levels towards their boundaries are eliminated with the aid of group or lateral entries, and are of lesser significance in the case of ventilation effected according to the diagonal scheme than in that of the retrograde scheme.

The *retreat method of mining* has the following advantages over the advance one: 1) working conditions in the entries are much better; 2) coal losses are smaller; 3) air leakages are eliminated; 4) fire hazards are reduced and fire-fighting is much easier; 5) preliminary driving of level entries allows a detailed complementary exploration of the structure and the mode of occurrence of the seam; 6) loading of coal brought to the haulage entry from stopes is simplified; 7) since the level is already developed by the driving of entries over its entire length, supplementary active stopes may be quickly made ready, if necessary.

It may thus be seen that retreat mining has important advantages over the advance one, in technical and, particularly, organisational terms. These advantages, however, are valid only for the retrograde scheme of ventilation and in the absence of group or lateral entries. The use of the latter tends to nullify the principal merits of retreat mining.

Hence, the question of relative advantages and disadvantages of these two methods employed in working mine fields appears to be extremely complex.

The chief drawback of the retreat method is that it requires preliminary driving of main entries along the entire length of the mine field flanks. This disadvantage, however, diminishes in direct proportion to the speed with which strike entries can be made. The smaller the length of the mine field flank and the greater the monthly rate of the face advance in the entry, the less time is required to prepare the level for retreat mining. In this connection of special importance are high-speed methods of driving development openings, based on rational use of a set of efficient machines and adequate organisation of work with a multicycle operation. The adoption of high-speed methods for driving mine workings has now been made possible by the serial manufacture of mining equipment, particularly of drifting machines and mine combines.

The more difficult the maintenance of level entries near the worked-out areas, the more advantageous the employment of retreat mining. Since unstable wall rocks and the growing thickness and pitch of the seams tend to increase these difficulties, it is preferable to use the retreat method in mining thin seams with moderate and high dip, with readily breakable coal and unstable wall rocks.

We have earlier mentioned the merits incident to the retreat mining of seams containing spontaneously combustible coal.

When there are apparent geological disturbances in the deposit, the information collected about them in driving development openings in the retreat mining of a level adds to the advantages of the method.

Because of the reasons cited above retreat mining was hitherto rarely practised, only in cases of real necessity.

But now, with the introduction of the method of high-speed driving of mine workings, this attitude should be revised. In many instances the reduction of the period of level development through high-speed driving may allow the use of the advantages inherent in the retreat working of mine fields.

The issue discussed above should not be confused with the question of mining coal blocks. It will be seen later, in Chapter XXII, that the extraction of blocks by the retreat method is practised quite frequently.

In mining the first level of a mine field, the nearest to the ground surface, it is the advance method that is always used, since there is no need to maintain the ventilating entry connected with the surface through air-pits over its entire length. If retreat mining is the dominant method in the mine, the operations may be organised in the following manner: to work the first level towards the boundaries of the mine field so as to start stoping as early as possible, and meanwhile develop the next level for its working by the retreat method.

## 14. Basic Principles of Estimating the Size of a Mine Field

If the size of a mine field does not depend on geological and certain other factors (see Section 15), it must be decided when the mine layout is planned. Let us briefly touch upon the *analytical method* used in estimating the size of the mine field at a mine with a given annual tonnage.

The gist of the analytical method is illustrated by a very simple example involving opening up a single gently pitching seam through inclined shafts (see Fig. 8). The object is to establish the size of the mine field that would ensure the minimal mining cost per one ton of coal excavated in the mine and charged against factors depending upon the size of the mine field.

To begin with, let us note that expenditure per ton of coal tends to diminish in one case, go up in another, remain unchanged in yet another—all that depending on the size of the mine field. One example of expenses in the first group is depreciation of capital outlays for the construction of surface structures of the mine: the larger the reserves of the mine, the smaller the part of the cost per one ton of coal extracted. The expenses in the second group are exemplified by the cost of haulage along the tramming entries, their maintenance, ventilation, hoisting of coal up the shafts. The bigger the mine field, the greater these expenses per one ton of the coal output. One example of expenses in the third group is the cost of stoping: the nature of operations performed at the active working faces remains essentially the same regardless of their location in the mine field.

To estimate the size of a mine field entailing minimal expenses per ton of coal charged against the variable costs of the first and second groups, let us first put down these expenses one by one for the entire mine field, add them up and divide the result by the total reserve tonnage of the mine field. This will give us the per-ton cost of coal output. In mathematical sense, this expression represents a function of the mine field size. Let us now find the size of a mine field which would correspond to the minimal per-ton cost of coal output, and that will furnish the answer to the problem.

Before proceeding with actual calculations, we should note that in this instance the inclined height of the level interval is determined according to formula (2)

$$h = \frac{A}{2Lpc}.$$

Let us denote the size of the mine field on the strike by $S$ and the number of levels sought by $n$.

Workable coal reserves in one level are denoted by $z$ and then the total reserves of the mine field will amount to

$$Z = nz.$$

If by $t$ we denote the number of years required to mine one level, the estimated service-life of the mine will be

$$T = nt.$$

Conformably to the above-mentioned designations, we will now put down the costs for the entire mine field according to items standing in direct relation to the size of the mine field.

1. *Shaft sinking.* Let us divide the overall length of the shafts into two separate sections: the upper portion, from the ground surface down to the upper boundary of the mine field, and the remaining, principal one, the extent of which is equal to the size of the mine field

down its dip, that is, $nh$. If the cost of the upper section (which, obviously, also includes the expenses incurred by special methods of timbering the shaft mouths) is designated by the sum total $K_s$ and the aggregate cost of one metre of inclined shafts and their manways by $k_s$, the overall cost of the shafts will be

$$K_s = nhk_s. \tag{5}$$

2. *Maintenance of shafts.* We shall disregard the cost of maintaining the upper portion of the shafts, since its length is insignificant, while actual maintenance in view of concrete or metal lining and support of shaft mouths requires but little attention. Denoting by $r_s$ the total maintenance cost per one metre of shafts and their manways, we find that the overall cost of maintaining the shafts for a period covering the mining of the first level, that is, during $t$ years and for the length of $h$, comes to $r_s ht$ (rubles). Similarly, we can estimate the cost of maintaining the shafts over the length $2h$ for the time interval $t$ required to work out the reserves of the second level. This cost may be put down at $2r_s ht$.

For the last, $n$-th level, we arrive, accordingly, at the outlay $nr_s ht$. The overall cost of maintaining shafts in the mine field will be

$$r_s ht + 2r_s ht + \cdots + n r_s ht = \frac{r_s ht\,(n+1)\,n}{2}. \tag{6}$$

3. The total outlay for the construction of surface mine structures, charged against the mine in question (excluding dwelling houses, which may be used not only by the mine concerned) we denote by

$$B. \tag{7}$$

4. By denoting the cost of one *shaft station* by $D$, we find that the aggregate cost of all shaft stations within the mine field will be

$$nD. \tag{8}$$

5. The cost of *driving strike entries* will amount to

$$k_e S\ n+1), \tag{9}$$

where $k_e$ is the cost of driving one metre of a level strike entry.

6. *The cost of maintaining level strike entries.* If any mine working has a permanent length of $l$ and must be maintained over $t$ years, the overall cost of its maintenance, with $r$ representing maintenance cost per one metre of its length, will equal

$$rlt.$$

But the level strike entries must be maintained in varying conditions, since their length either gradually increases (with the advance mining of the level) or decreases (in retreat mining).

Hence, if the length of the strike entry varies from 0 to $l$ in the space of $t$ years, its mean length kept up during this period will be $\frac{l}{2}$ and, therefore, the cost of maintaining such an opening in these conditions will be

$$\frac{rlt}{2}.$$

Taking this expression to represent the cost of maintaining strike levels and bearing in mind that the mine field has two flanks, each $\frac{S}{2}$ metres long, that over the period of $t$ years two level entries have to be maintained in each level (haulage and ventilating) and that the total number of levels amounts to $n$, we find that the total maintenance cost of all strike entries in the entire mine field is

$$\frac{r_e \frac{S}{2} t}{2} \times 2 \times 2n = nr_e St \tag{10}$$

where $r_e$ is the cost of maintaining one metre of strike entry per year.

7. *The cost of hoisting coal up the shafts.* If any load $Q$ is to be conveyed over distance $l$, the "performance" of transport facilities is expressed by $Ql$ (ton metres, ton kilometres). Denoting the cost of one ton metre by $q$, we find the overall haulage cost in this particular instance is $Qlq$.

Overall transport cost $q$, however, depends upon distance $l$. It may be shown that

$$q = \frac{q_1}{l} + q_2,$$

where $q_1$ and $q_2$ represent certain parameters, whose significance and numerical values depend on the type of haulage used in the mine.

Consequently, the cost charged against one ton metre is composed of two addends, one related to distance, the other not.

With the foregoing in mind, we now proceed to estimate the cost of hoisting coal up the shafts for the entire mine field (ignoring the insignificant hoisting cost along the upper section of the shaft, from the top boundary of the mine field to the shaft mouth):

$$zh\left(\frac{q_1}{h} + q_2\right) + z2h\left(\frac{q_1}{2h} + q_2\right) + \cdots + znh\left(\frac{q_1}{nh} + q_2\right) = Zq_1 + \\ + zhq_2 \frac{(n+1)n}{2}. \tag{11}$$

8. *The cost of coal haulage along level entries.* It is not difficult to prove that this cost is

$$nz \frac{S}{4} q^e = Zq_1^e + \frac{nzSq_2^e}{4}. \tag{12}$$

9. To calculate the cost of *hoisting men* in inclined shafts, let us introduce a denotation $q^m$, which is the cost of transporting men in relation to 1 ton metre of work performed in hoisting the load. The total cost of man-hoisting in an inclined shaft will then be found to be

$$Zq_1^m + zhq_2^m \frac{(n+1)\,n}{2}. \tag{13}$$

10. The cost of the *man-riding mechanised haulage along level entries* may be arrived at in similar manner:

$$Zq_1^{me} + \frac{nzSq^{me}}{4} \tag{14}$$

The cost of ventilation and mine drainage by means of pumping may be estimated for the entire mine field, but we shall refrain from doing so, since inclusion of these factors into our estimates will exert a certain influence on the size of the mine field (towards its reduction) only in instances of highly gaseous and watery mines.

If we now add up all these individual expenses and divide the total by the workable reserve tonnage in the shaft field, determined by one of the following expressions

$$Z = nz = Atn = \frac{ASn}{2L}, \tag{15}$$

the formula representing the level of expenditure depending on the size of the mine field and falling on to 1 ton of the coal produced will, following pertinent algebraic transformations, assume the form of a function of two unknowns, viz., the size of the mine field along the strike and the number of levels $h$ in the mine field:

$$f(S, n) = c_1 S + \frac{c_2}{S} + c_3 n + \frac{c_4}{n} + \frac{c_5}{Sn} + c_6, \tag{I}$$

where

$$c_1 = \frac{r_e}{A} + \frac{q_2^e + q_2^{me}}{4}; \tag{16}$$

$$c_2 = \frac{2L}{A}(k_s h + D); \tag{17}$$

$$c_3 = \frac{h}{2}\left(\frac{r_s}{A} + q_2^s + q_2^{sm}\right); \tag{18}$$

$$c_4 = \frac{2Lk_e}{A}; \tag{19}$$

$$c_5 = \frac{2L}{A}(k_s + P). \tag{20}$$

The value of the addend $c_6$ is not indicated, since, as it will presently be seen, it has no bearing on the solution of the problem.

Let us find the values of the mine field size on the strike and the number of levels favouring minimal expenditure per ton of output. This means establishing values $S$ and $n$ under which function (1) reaches its minimum.

*Notes:* 1. By its physical nature the variable $n$ is integral, but content with the approximate solution of the problem we face, with the accuracy known in advance to be sufficient to allow proper planning, we shall regard value $n$ as constantly variable.

2. The single-valued (unambiguous) existence of the $f(S, n)$ minimum is made clear by the physical nature of the problem, and there is no need to resort to mathematical investigation to prove it formally.

To determine values $S$ and $n$ which reduce $f(S, n)$ to its minimum, it suffices jointly to solve the equations:

$$\frac{\partial f}{\partial S}=0; \quad \frac{\partial f}{\partial n}=0;$$

or represented in detail:

$$c_1-\frac{c_2}{S^2}-\frac{c_5}{nS^2}=0; \tag{II}$$

$$c_3-\frac{c_4}{n^2}-\frac{c_5}{Sn^2}=0. \tag{III}$$

The graphic method appears to be the simplest way of achieving joint solution of these equations.

Having found the optimal size of the mine field, we can calculate the tonnage of workable coal reserves contained therein. By dividing these reserves by the annual output of the mine, we find its planned service-life.

*Numerical example.* The object is to determine the size of a mine field worked by an inclined shaft. The extractable deposit comprises a single gently pitching seam 1.5 metres thick, the annual output of the mine $A=300{,}000$ tons.

Having found $p=1.8$ ton/sq m; $c=0.9$; $L=400$ metres, we determine by formula (2) that $h=230$ metres.

Let us then accept the following values for the parameters included in the solution of the problem: $K_s=350{,}000$ rubles; $k_s=2{,}900$ rubles; $r_s=70$ rubles; $B=3{,}500{,}000$ rubles; $D=400{,}000$ rubles; $k_e=150$ rubles; $r_e=80$ rubles; $q_2^s=0.0003$ ruble per ton metre; $q_2^e=0.0001$ ruble per ton metre; $q_2^{sm}=0.00015$ ruble; $q_2^{me}=0.0001$ ruble.

With these values for the parameters above, the calculations according to formulas (16)-(20) will result in: $c_1=0.00032$; $c_2=2{,}850$; $c_3=0.0782$; $c_4=0.4$; $c_5=9{,}650$.

In order to solve equations (II) and (III) graphically, let us determine the $S$ values for each:

$$S=\sqrt{\frac{c_2}{c_1}+\frac{c_5}{nc_1}} \tag{IV}$$

$$S=\frac{c_5}{c_3n^2-c_4} \tag{V}$$

and then, by assuming consecutive values $n=1$; $n=2$; $n=3$ ..., let us plot corresponding curves. Their intersection point will give the optimal values for the size of the mine field and the number of levels we seek. Inasmuch as the number of levels found by this procedure will, generally speaking, represent a fraction, we will ultimately have to accept the closest integral number for our mine layout.

In the present case, we find $S=6{,}300$ metres; $n=3.7$. Ultimately, in round numbers, we adopt $S_0=6{,}000$ metres; $n_0=4$.

Hence, the workable reserves of the mine field will be $Z=Snhpc=8.9$ million tons and the service-life of the mine

$$T=\frac{Z}{A}=30 \text{ years.}$$

A few observations ought to be made with reference to the above-discussed method of estimating the size of the mine field.

In order to arrive at a proper solution, particular attention should be paid to very accurate and substantiated selection of numerical values of the parameters included in the problem.

The final values for the size of the mine field thus obtained should be regarded as approximate, since, firstly, the accuracy of calculation itself is rather limited and, secondly, even theoretically, the values of arguments $S$ and $n$ within the range of the function $f(S, n)$ minimum may be made to change rather significantly without markedly affecting the value of the function $f(S, n)$ proper, that is, altering the level of expenditure dependent on the mine field size per ton of output.

The solution of the problem does not depend on absolute cost values, but on relation between them, since each value of $C_i$ includes the cost only once, while the solutions of the problem, in the final analysis, are determined by equations (IV) and (V), which include only *ratios* of the $C_i$ value. In other words, general price changes do not influence the solution of the problem.

## 15. Annual Output and Service-Life of Coal Mines

Information given above makes it possible to discuss the major problem of establishing the annual output and service-life of mines in greater detail.

In Chapter I (Section 8) it was stressed that mine layouts and operations in conditions prevailing in the Soviet socialist economy should be so planned as to make mining enterprises meet to the fullest the requirements of the national economy. The types of mines, their annual production capacity, service-life, equipment and surface structures must, in particular, ensure high labour efficiency and low production costs.

In establishing the annual output of mines, their service-life and the corresponding size and reserves of mine fields, we are bound to encounter two characteristic types of coal deposits:

1) deposits broken by geological disturbances—folds (Fig. 41), faults (see Fig. 5), pinching outs (see Fig. 6) and others—into isolated sections, each known in advance to be worked by no more than one mine. Consequently, the reserves, size and boundaries of the mine field are in this instance quite definite;

2) deposits spreading continuously over large areas and known in advance to be worked by several mines, this making it essential for the planners themselves to decide the size, boundaries and reserves of the mine field of each of these mines, since these magnitudes are not limited by natural factors.

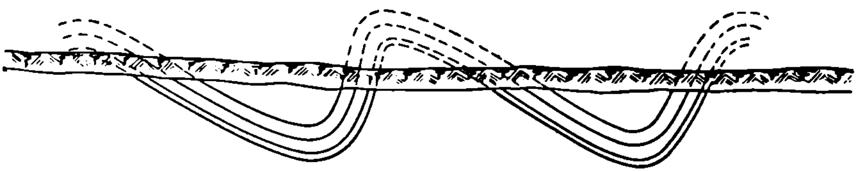


*Fig. 41.* Folded deposit with sections allotted for individual mines

Let us call the sections of the first group of deposits *limited* in reserves and those of the second—*illimited.*

For limited sections the workable reserves of the mine field are fixed, that is, $Z=\text{const}$.

Since there is an obvious correlation $AT=Z$ between the annual output of mine $A$, its estimated service-life $T$ and workable reserves of its field, and, since for sections with limited reserves $Z=\text{const}$, there exists an inverse relation between the annual output and service-life of the mine in this instance. For example, with $Z=5$ million tons, the above-mentioned values will vary as follows:

| $A$ (thousand tons) | $T$ (years) |
|---|---|
| 100 | 50 |
| 200 | 25 |
| 500 | 10 |
| 1,000 | 5 |

It may be said that there exist numerical values of annual output $A$ and the service-life of a mine under which the per-ton cost of output charged against items dependent on $A$ and $T$ will be minimal.

As a matter of fact, the higher the annual production of a mine, the greater is the capital invested in its construction. It is true that these outlays do not rise proportionately to the increase of the annual production capacity of a mine, but at a slower rate. In order to arrive at the depreciation rate of these capital investments per ton of output, it is necessary to divide them by the amount of workable reserves of mine field $Z$, which in this case is constant. The nature of relation between the depreciation rate per ton of output and other factors will be determined by curve $a$, shown in Fig. 42.

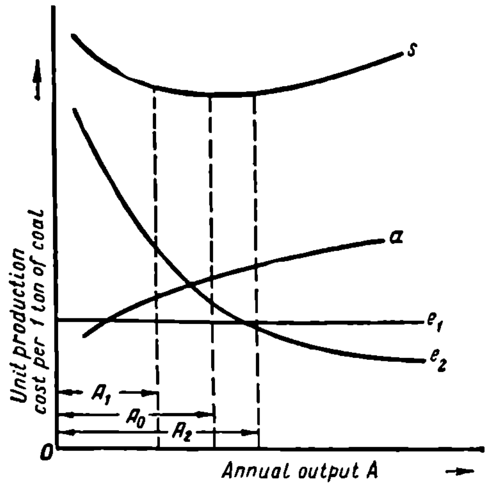



*Fig. 42.* Relationship between the cost of coal mining and the annual output of a mine in sections with limited reserves

Operational expenses per ton of output may be broken into two separate groups. A portion of them (for instance, for stoping operations) will not depend upon the annual production of the mine and in Fig. 42 it is represented by curve $e_1$, running parallel to the horizontal axis. The other portion of the expenses will decrease with the rise of $A$. The costs included in this group are largely associated with the service-life of mines with different production capacity required for the extraction of the same mineral reserves. For example, with $A = 100{,}000$ tons, labour costs charged against hoisting loads up the shaft would continue for 50 years, while with $A = 500{,}000$ tons—for only 10 years, although in both these periods the amount of coal taken out of the mine would be the same, viz., 5 million tons. In Fig. 42 the expenses are depicted as curve $e_2$. In order to plot summary curve $s$ for the above-cited costs, it is necessary to add up the ordinates of all the three curves. The shape of curve $s$ may furnish grounds for important inferences:

1. Since curve $s$ has a minimum, it means that, in building mines on sections with limited reserves, it is possible to find annual mine output $A_0$ under which we can achieve the minimal cost of coal production.

2. There is an optimal period of service-life for any mine with a given production capacity:

$$T = \frac{Z}{A_0}.$$

Consequently, the service-life of a mine is not determined, as it is sometimes claimed, by the "normal" depreciation of initial permanent installations and equipment (at the mines these include a wide variety of objects with different depreciation rates), but is dependent on the optimal combination of depreciation costs and running expenses per ton of output. If, because of the mine's short service-life, some transportable object (say a machine) is not fully depreciated, it can be dismantled after completing the extraction of a particular mine field, and used at another mine. The design of some structures allows them to be made sectional (headframes, for instance).

3. The smaller the workable reserves of a mine section, the shorter its service-life and the lower its production capacity.

4. It is of utmost importance to note that curve *s* changes very smoothly. This means that, though the point of its minimal value is in fact indicative of a definite optimal annual capacity of mine $A_0$, certain deviations from this figure almost do not affect the per-ton cost of output. In other words, if some lower capacity $A_1$ or a higher one—$A_2$, is accepted instead of $A_0$, this will fail appreciably to alter the prime cost of the mineral extracted.

Hence, the inference that the optimal annual output of a mine should be expressed not by a strictly definite figure but by an approximate range of figures: "from-to". This important observation refers also to the service-life of mines. In the planning of the layout and operation of mines, the above-mentioned features make it possible appreciably to vary the estimated values, this depending on other considerations.

In the determination of the annual production capacity and service-life of mines working deposits with "illimited" reserves present a problem of far greater complexity. It is possible, however, to prove that in this instance too there exist optimal figures for these values and a region adjacent to them in which these values can vary without noticeably influencing the per-ton cost of output. Generally speaking, estimates show that the construction of big mines on rich dep s-its is economically advantageous. The annual capacity of such mines, however, should not exceed certain limits. Megalomania in this respect would result in many harmful effects, such as increased production cost, unjustifiably long construction period and organisational difficulties in the management of an unduly big mine.

Since in steeply pitching deposits the height of level intervals is limited to a rather narrow range (see Section 8), the annual output of a mine may, in such conditions, be determined in accordance with formulas (1) and (3). For instance, in conditions prevailing in the Donets coal fields, fairly characteristic are the following figures for the values in formula (3): $L=400$ metres; $h=150$ metres; $k=0.8$; $\Sigma p=12$ tons per sq m; $c=0.9$. Hence, $A=1{,}040{,}000$ tons, or roundly—1,000,000 tons per annum.

In the mining of steeply dipping deposits, extending to a great depth, the mines may exist for many decades, going deeper with each consecutive level. The reconstruction of surface structures and plant is carried out steadily during this period. Inasmuch as with the progress of mining operations to deeper levels the abundance of gas in the mine, as a rule, tends to increase, it becomes necessary to sink additional air shafts and make other mine openings. One example is the deep mines of the main anticline in the Donets coal fields.

It thus follows that the annual output and service-life of coal mines vary substantially.

At present the principal types of coal mines are those with daily output of 1,000, 1,500, 2,000, 3,000, 4,000 and 5,000 tons. Since the number of workdays in planning mine operations is set at 300 per annum, the foregoing daily production figures correspond to annual output of 0.3, 0.45, 0.6, 0.9, 1.2 and 1.5 million tons. Still bigger mines can be planned for working deposits with greater geological reserves and better conditions of occurrence. On the other hand, for sections with limited geological reserves and faulted seams, it is better to design mines with a daily estimated output of less than 1,000 tons.

The service-life of mines is directly related to their production capacity. When the daily output comes to 1,000 or 1,500 tons, the service-life of a mine should not be less than 30 years; when output is 2,000 tons—not less than 40 years; when production comes to 3,000 or 4.000 tons—not less than 50 years. In the case of mines with a daily capacity exceeding 4,000 tons the service-life should not be less than 60 years.

It is the author's opinion that rich deposits with "illimited" reserves, dipping at low angles, should be worked by big mines with an approximate annual capacity of up to 1.5 million tons and service-life of about 30-40 years. Longer service-life is possible in the case of mines working heavy pitched deposits, provided they are rebuilt during the period of their service. The duration of service-life for mines with an annual capacity ranging from 300,000 to 600,000 tons is from 20 to 30 years.

For mines operating in sections with "limited" reserves the periods of service-life are still shorter; at mines with annual output of 400,000-500,000 tons, it may be set at 10-15 years. For sections with "limited" reserves the annual production capacity of mine $A$, its estimated service-life $T$ and the workable reserves of the mine field $Z$ should be coordinated so as to comply with correlation

$$Z = AT.$$

*Productive-exploration mines*, set up with the view to a detailed study of conditions attending the working of a deposit and properties specific to the mineral, exist only a few years and their annual capacity is insignificant.

## 16. Sequence of Mine Fields

Generally, depending on the nature of the geological structure of coal deposits, it is not one but several and sometimes scores of mines that are built simultaneously or one next to another in any particular region. Their location and the boundaries of the adjoining mine fields must be well coordinated.

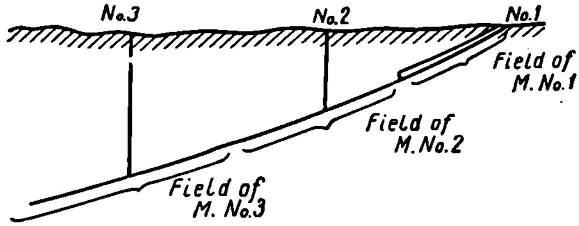



*Fig. 43.* Location of mine fields down the dip

When mining individual gently pitching beds, the shafts are sunk along the line running down the dip in the order of Nos. *1*, *2*, *3* (Fig. 43), that is, with the shallower shafts coming first.

Firstly, this sequence allows quicker stoping in the deposit. Secondly, operation of the shallower shafts of the first round makes it possible to investigate the deposit in all its details, particularly with respect to the location of geological irregularities, this creating a basis for confidently deciding on problems of early development and for making a proper choice of mining methods in planning and building subsequent, deeper shafts.

Shallower shafts may have mine fields of smaller size, both on the line down the dip (Fig. 43) and along the strike (Fig. 44).

The latter drawing is schematic in that it illustrates only the extension of mine field areas downward, without defining their relative size.

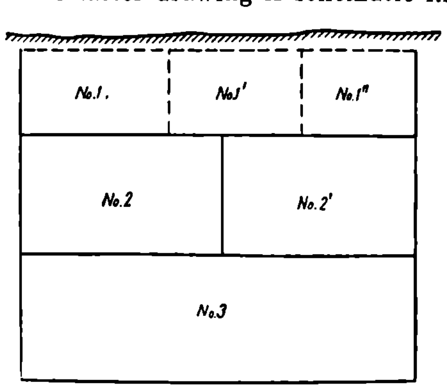



*Fig. 44.* A diagram showing the position of adjacent mine fields

Note should be taken of the fact that in thoroughly prospected and well-explored deposits it is a frequent practice at once to put down big shafts to work beds right from their show beneath the overburden.

In the mining of coal measures, mine fields should be well connected not only on their dip and strike, but across the strike too. For example, to work coal measures at shallow depth individual shafts may be sunk, while at deeper levels these same seams can be opened up jointly (Fig. 45).

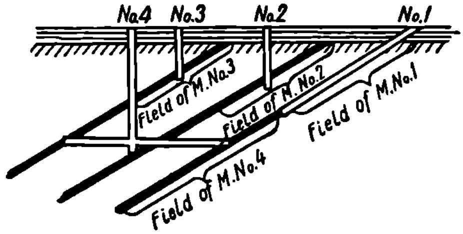



*Fig. 45.* A diagram showing the arrangement of mine fields in working a coal measure

Mines constituting the first part of the project may include inclined shafts, while those of subsequent rounds can be worked by vertical ones (see Figs 43 and 45).

Sometimes in exploiting a mine it is possible to utilise the abandoned workings of neighbouring mines. Thus, for mine No. *2* (see Fig. 44) the deepest haulage entry of mine No. *1* can be made to serve as the uppermost airway, while the workings in the field of mine No. *3* can be ventilated through the old shafts of mine No. *2*. It is precisely in such instances that boundary ventilation is used for a mine field (see Section 7), inasmuch as by the time the development of a new field is started through a deeper mine, there are shafts and other openings available in the old mine field, allowing this scheme of air circulation.

It should be borne in mind that newly developed mine fields may be overlaid up their dip by abandoned mined-out areas of other mines that are liable to be *flooded*. Therefore, when the workings of, say, mine No. *2* approach the abandoned areas measures must be taken to prevent inrushes of mine water. The most effective is the preliminary *pumping out* of water through the shafts of mine No. *1*, this making the advance operations of mine No. *2* absolutely secure. Should the pumping operation be rendered impossible, for example, by the uncontrollable caving-in of the roof rocks in old workings, the water may be *drained* down to the collector and pumping plant of mine No. *2*. The water is discharged through specially equipped boreholes drilled upward from the faces of the deeper mine workings to the flooded areas. Water inrush hazards also exist when there are flooded areas in the field of the adjacent mine extending on the strike, or when the workings of the overlying bed running immediately above the productive seam are flooded (see Fig. 45). In such cases, it is also common preliminarily to pump out the water or discharge it through special boreholes.

## 17. Preparation of Complex Plans and Layouts for Mining Mineral Deposits

The interrelation of neighbouring mines with regard to the location of mine openings and some of the surface structures is so close that, before getting down to planning the layout and exploitation of a deposit, it is essential to have at least a rough preliminary idea about the location of mines and mine fields over the entire explored area of the deposit. Since the fields of adjacent mines are contiguous, some workings may be utilised to serve two of them. One illustrative example is the use of old shafts and permanent inclines for boundary ventilation in an underlying mine field, as said above. Transport facilities of all kinds and types—railways, highways and cableways—should be situated so as to be of maximal convenience and accessibility for the whole group of mines. The same holds true with respect to the location of water supply lines, power lines and, in some cases, of compressed-air lines. Industrial plants, such as dressing mills, coke ovens, machine shops, power substations, etc., as well as dwellings, cultural and communal institutions, clubs, hospitals, etc., may be built to serve two or more mines at a time. To speed up the construction of mines, minimise their cost and simplify their operation, no effort should be spared to make their equipment and plant as standard and uniform as possible.

In planning the working of a deposit, it is not only in space that issues relative to the location and construction of mines are to be considered, but also in time, that is, with respect to the *sequence* in which mines and surface installations should be built over the years.

From the above it follows that mines must be designed as a *complex set*. All basic issues should be considered in their relation not only to a single industrial unit but, as a rule, to their aggregate set, which includes all the industrial, administrative, supply, dwelling and cultural and communal buildings and structures.

This idea acquires special importance in the case of new areas, where mining enterprises are built virtually from scratch.

The possibility and advisability of complex planning of mining and other industrial enterprises in the U.S.S.R. are inherent in the very nature of the Soviet national economy, an economy based on principles of socialist planning.

Before plans and layout blueprints for individual mines and open-cuts have been prepared, a *long-term project for the industrial exploitation of the whole new deposit* should be elaborated and approved in each individual case. The purpose of such a project is to assure the most expedient opening of the entire deposit and its division into individual mine fields, as well as to take a rational decision on problems of coal dressing, out-of-mine transportation, electric power and water supply, location of dwellings for mine workers and organisation of construction work in conjunction with the general plan for the opening and exploitation of the deposit.

## 18. Notion of Alternative Methods or Versions in Planning Opening and Development of Deposits

It was repeatedly said above that the advisability of adopting any method of opening as a whole or in part may be verified by comparative technical and economic calculation proving the expediency of all possible alternatives. The procedure of elaborating and comparing alternative versions plays a major role in planning the operation of mining and industrial enterprises in general.

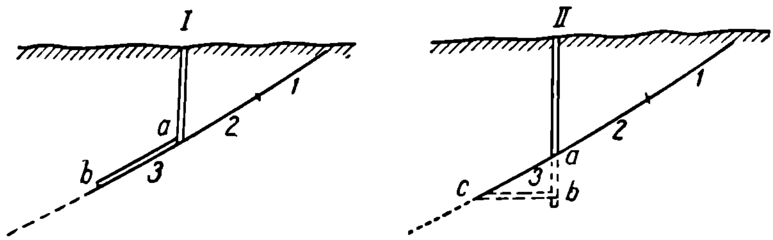



*Fig. 46.* Alternative methods of opening a mine

The substance of this procedure may be illustrated by the following simple example. A vertical shaft has been sunk to mine an inclined bed (Fig. 46) and two upper levels have already been worked out. To proceed with the development of the third level, there are two alternative ways possible:

1) to drive from shaft station *a* permanent incline *ab* and through this extract the available coal reserves of the third level (Fig. 46, *I*). To the surface coal will be hoisted up from shaft station *a*;

2) to deepen the shaft vertically down to the third level over distance *ab* (Fig. 46, *II*), to drive and equip shaft station *b* and then make crosscut *bc*. Coal will be brought to the surface from shaft bottom *b*.

To form a judgment of the economic expediency of the I and II alternatives, it is obviously necessary to determine the technical differences distinguishing them, and then to compare the costs, which will be unequal in the two versions.

Alternative I provides for:

a) driving and equipping a permanent incline with a parallel manway with hoisting plants for men and loads;

b) maintaining this opening for the period necessary to extract the reserves of the third level;

c) bearing running costs incident to the hoisting of loads up the permanent incline;

d) installing an auxiliary pumping plant on level *b* to drain mine waters from this level to shaft level *a* and continue operation of the pumping unit formerly set up at the last level.

Alternative II entails:

a) deepening the shaft over distance *ab*;

b) making arrangements for shaft station *b*;

c) driving crosscut *bc*;

d) maintaining this crosscut for the duration of mining operations on the third level;

e) bearing the haulage costs charged against transporting the reserves of the third level along the crosscut. However, it should be noted that, as distinct from alternative I, alternative II does not provide for any transfer of loads from one type of transport to another, since the trains running along the entries reach shaft station *b* directly via the crosscut;

f) setting up a pumping station at shaft station *b*;

g) bearing additional expenses incident to the hoisting of loads up the shaft over distance *ba*.

After the preparation of schematic projects for both alternatives, all the expenditure incurred within a definite period of time, say, a year, is added up. The result is included in a comparative table for the alternatives (Table 1).

In the instance under review, Table 1 envisages the following: annual output of the mine—200,000 metric tons; time limit set for mining the level—5 years, inclined height of the level interval—200 metres; workable reserves of coal in the level—1 million tons.

Each item of expenditure is calculated separately; the table carries only the result. In compiling Table 1, it was deemed convenient (though not essential) to take for purposes of calculation five years as the duration of the service-life of the level. The cost items included in this table cannot be used for reference purposes, since they are intended merely to illustrate the mode and procedure to be followed in such estimates.

The ultimate figures in the table show that the difference in favour of alternative II equals 562,000 — 454,300 = 107,700 rubles, or

$$\frac{107.7}{454.3} \times 100 = 23.8 \text{ per cent}$$

of the smaller of two totals.

*Table 1*

**Tabulated Comparison of the Alternatives**

| Items of expenditure | Cost, in thousand rubles | |
|---|---|---|
| | Alternative I | Alternative II |
| *Excavation of workings and openings* | | |
| Shafts | — | 200 |
| Shaft stations | 150 | 200 |
| Crosscuts | — | 50 |
| Permanent inclines | 60 | — |
| Permanent incline chambers | 12 | — |
| *Maintenance of workings* | | |
| Permanent inclines | 20 | — |
| Crosscuts | — | 4.3 |
| *Mine hoisting* | | |
| Labour cost at the shaft station | same | same |
| Labour cost at the permanent incline | 210 | — |
| Mechanical equipment and plant of the inclines | 25 | — |
| Electric power | same | same |
| *Mine drainage* | | |
| Labour cost | 35 | — |
| Electric power | same | same |
| Transport of men | 50 | — |
| Ventilation | disregarded | disregarded |
| Total | 562 | 454.3 |

Hence, the alternative of deepening the vertical shaft and driving a crosscut is more economical and, besides, more convenient for the conduct of mining operations.

If the mine field were divided into four, instead of three, levels (dash line in Fig. 46), it would be necessary, while discussing the problem of the best alternative for the development of the third level, to take account of its possible effect in the future on the opening of the fourth level. In other words, in compiling a summary table of costs, it would be indispensable to consider methods for combined development of both the third and fourth levels.

Although extremely simple, the application of the method of alternatives in planning work requires adherence to the following definite rules to avoid errors in estimates:

1. Before proceeding with an economic comparison of the costs involved in the alternatives, it is necessary to consider them carefully from the technical standpoint in order not to omit any costs incident to any one of them. This rule is of paramount importance inasmuch as economic comparison of the alternatives, made without first exhaustively clarifying all their technical aspects, usually leads only to erroneous conclusions.

2. Only important costs should be taken into account. In Table 1, for example, ventilation costs were dropped, since the only difference in this respect is that in one alternative the air current flows down the permanent incline, while in the other along the shaft and cross-cut.

3. The gauge of the importance of the costs is not their absolute value, but rather the relative one. When the total comparable sums run into tens of millions of rubles, expenses of tens of thousands of rubles may be disregarded; on the other hand, in a total amounting to a few hundred thousand rubles they are quite significant.

4. Only different costs are compared; numerically identical expenses are left out.

5. Slightly different costs are considered identical and excluded from the comparison.

6. Comparison of operation and depreciation costs requires that all outlays refer to some definite time interval—day, year, etc.

7. If essential, separate *subalternatives* should preliminarily be prepared for individual sections of the alternatives and compared with each other. Thus, in the example referred to above (see Fig. 46) the following subalternatives could be considered for mine drainage: 1) near the permanent incline water from level $b$ is delivered by an auxiliary pumping unit to level $a$, and thence to the ground surface by earlier installed pumps; 2) near the permanent incline a sufficiently powerful pumping plant, capable of delivering mine water to the surface at once, is set up on level $b$, while the pumping unit formerly operating on level $a$ is closed down; 3) similarly, in the case of alternative II an auxiliary pumping station is set up at shaft station $b$ to deliver water to level $a$, while the pumps available on this level continue their operation, or 4) a powerful pumping plant is set up at shaft station $b$ to deliver mine water directly to the surface. Such subalternatives are compared individually and the best is taken into account when comparing the basic alternatives.

8. The difference in the cost of the alternatives is expressed in per cent of the lower of the totals under comparison.

9. Estimates made for comparing alternative projects for the opening up of mine deposits are commonly taken to be accurate within 10 per cent.

10. Therefore, if the expenditure entailed by one alternative does not exceed the other by more than 10 per cent, the alternatives are considered economically equivalent and preference is given to the one which is technically superior. For example, alternative II with the crosscut is more suitable for operation than alternative I with its permanent incline.

The degree of accuracy of the estimates used in comparing the alternatives may, however, be raised by enhancing the thoroughness of calculation and by a more detailed consideration of individual cost items. To find proper solution for basic problems relating to the opening up and development of mineral deposits, it suffices to have *gross* cost figures.

11. In preparing and comparing alternative methods of developing and mining mineral deposits, special attention is paid to *distribution of jobs and to workers' pay* since this item of cost is decisive in mining, which is a *labour-consuming* industry.

The ultimate choice of alternative should be made with due consideration of the time needed for the realisation of each alternative under comparison.

## 19. Some Observations on the Opening up of Noncoal Bedded Deposits

Because of their sedimentary geological origin, some occurrences of nonmetal minerals, other than coal, may be bedded or sheetlike in shape. This category includes deposits of combustible shists, rock salt, potassium, phosphorites, gypsum and some other useful minerals. Since it is the *shape* of the deposit that primarily influences the choice of the method of opening and developing the beds or sheetlike occurrences of the minerals listed above, they are opened up in the absolutely same manner as that employed in the instance of coal beds; hence, all that has been said in this chapter applies to them too.

CHAPTER III

# OPENING UP OF ORE DEPOSITS

## 1. Shapes of Ore Occurrences

Because of the variable nature of the geological origin of metalliferous deposits, the shape and size of *ore bodies* may differ very widely.

On account of their sedimentary origin, some ore deposits occur in the shape of *regular beds* or *blanket formations.*

Typical of bedded ore deposits are the occurrences of manganese ore in the Nikopol (Ukraine) and Chiaturi (Georgian Republic) areas. Ore beds in these areas are approximately flat.

Some iron ore deposits in Krivoi Rog are similar to beds in shape.

Many ore bodies in the same Krivoi Rog basin occur in the form of typical sheetlike deposits. Most of them are distinguished by their high dip.

One example of small flat-lying bedded deposits is the iron ore occurrences in the Central regions of the U.S.S.R. In the horizontal projection they have the form of ovals with irregular contours several hundred metres long. Their thickness ranges from 0.25 to 3.5 metres, with ore reserves in some bodies amounting to hundreds of thousands of tons, and, though rarely, to 1 million tons. Steep dips are, as a rule, characteristic of the occurrences of chalcopyrite and pyrite *lenses* in the Urals (lenticular ore bodies of irregular shape).

An important type of *placer deposits* are *alluvial* or *river* placers. They are formed by particles of heavy metals or their ores washed by water streams. These particles accumulate in the lower layer of mineral sediments—silt, sand, coarse gravel, etc. In its horizontal projection, therefore, the alluvial placer follows the outline of sediments deposed by the water stream which engenders it. *Residual* placers, that is, those formed in the weathering zone of a primary (usually lode) deposit and not subjected to any appreciable migration from the site of their origin, are, as a rule, of secondary industrial importance, although they serve as an important factor in the discovery of primary deposits. The placers contain gold, platinum, platinoids (for example, osmiridium), as well as tin and tungsten ores.

The ores of many metals, especially nonferrous and rare, occur in the shape of *lodes*. Because of their geological origin (an ore vein is a mineral formation of ore or ore-containing minerals filling a *fissure* in the earth's crust), such veins often dip at high angles. Their thickness varies from a few millimetres to many metres. Disturbed occurrence in the form of "bulges", attenuations, pinches, etc., is common more to lodes than beds. The ore stuff is either localised in the vein proper or dispersed in the rocks enclosing the vein in the form of *disseminated* ore. The outlines and size of a vein in the plane of a fissure are quite variable, but in most instances the contours of the pinching out veins assume the shape of irregular, intricately curved lines. When the long axis of a vein extending down does not coincide with the dip of the deposit but forms an angle with it, this is designated as *pitch* of vein (or, for that matter, of any ore body). Veins formed by the ore stuff filling a series of closely lying fissures are said to be *composite* or *multiple*. Very often veins occur in systems or series, and not singly. In the U.S.S.R. many gold, silver, lead, tin, mercury, tungsten, molybdenum and other deposits occur in the shape of lodes or veins. Sometimes a vein contains more than one useful component as in the instance of silver-lead lodes.

At times ore occurrences are shaped like bodies in which all the three dimensions are more or less the same, although the general shape may be utterly irregular. Such ore bodies, when they are of large size and occur close to the surface, are usually mined by the open-cut method. One example is iron-ore deposits of the Magnitnaya Mountain (Southern Urals), Blagodat Mountain (Central Urals), the so-called porphyritic copper ores of the Kounrad deposit (Kazakhstan, in the vicinity of Lake Balkhash) and others.

Ore bodies of singular shapes form when karst cavities in limestones fill with ore substance (some bauxite deposits).

Ore beds, deposits, veins, and ore bodies in general, may outcrop with only the most recent drift beds covering them, or else, peter out and fail to reach the ground surface altogether. In the latter case, they are said to be *blind*. When preparing a project for opening up a deposit, one should take these blind ore bodies into consideration. Their presence is established by prospecting and exploratory work.

Early development of ore deposits is conducted by diverse methods, this depending on their shape, size and angle of dip, depth of occurrence and surface topography. Sections 2-5 below are devoted to the discussion of typical examples illustrating the opening up of ore deposits. Opening by *adits* employed in mountainous regions is described in brief in Chapter IV, Section 7.

## 2. Opening up of Gently Dipping Ore Deposits

Gently sloping or flat ore beds are developed by methods analogous to those enumerated above in describing the opening of coal seams.

The flat, shallow-occurring manganese ores in the Nikopol area are thus opened through vertical shafts. The ore beds there are 1-3.5 metres thick. The enclosing country rocks are soft and for that reason the mine fields, which are rectangular with sides hundreds of metres long, are worked by retreating methods.

The accepted standard pattern for opening up and developing mine fields is shown in Fig. 47. To avoid ore losses in safety pillars, hoisting and ventilating shafts are sunk outside the mine field boundaries.

Main drifts are pushed forward from the shafts, bisecting the mine field lengthwise. From these main drifts, starting in the centre and near the lower boundary of the field, twin panel drifts are driven, from which extraction drifts are made from the field boundaries.

On account of the marked montain relief of the land in the Chiaturi area flat manganese ore beds there are opened up through adits.

The method adopted for opening up small sheetlike iron ore deposits in the Lipetsk area (Voronezh Region), occurring at a depth of 20-25 metres, is illustrated in Fig. 48. Vertical shafts are sunk down to the ore body and are then connected by a breakthrough, this securing two escape-openings to the surface, ensuring adequate ventilation of underground workings and enabling to proceed with the making of development and, subsequently, production openings.

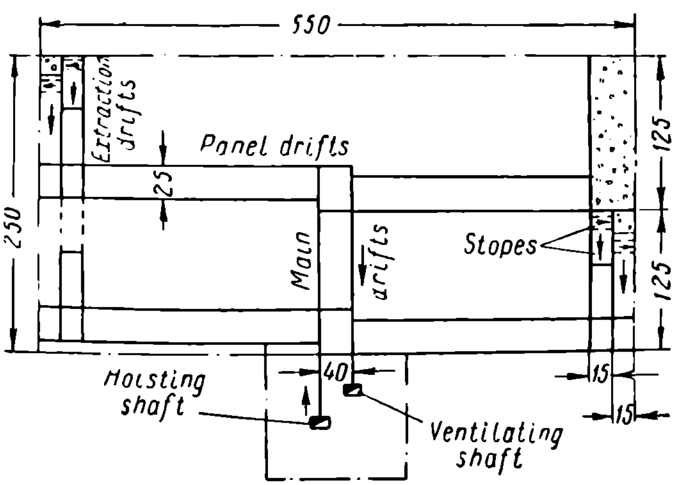



*Fig. 47.* Opening up of a manganese ore bed

A nonuniform foot wall of the deposit can be developed through workings driven in country rocks (Fig. 49).

Slightly dipping copper ore deposits in Dzhezkazgan (Kazakhstan) have a rather irregular outline in the horizontal plane. They are opened through vertical shafts (Fig. 50), from which crosscuts are driven to the ore bodies. The mined-out areas look singularly peculiar, since support pillars of ore are left to ensure the stability of the roof rocks of the deposit (for details see Chapter XXI, Section 2).

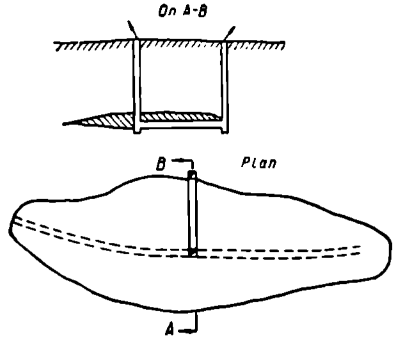



*Fig. 48.* Opening up of a sheetlike iron ore deposit

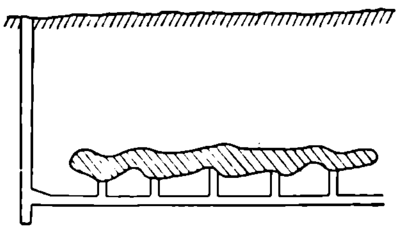


*Fig. 49.* Opening up of a deposit through mine workings driven in country rock

*Placer* deposits ordinarily occur quite close to the surface and, therefore, are often mined by open method. However, should *underground mining* prove more economical, the placer is divided into small-sized mine fields because of the soft enclosing

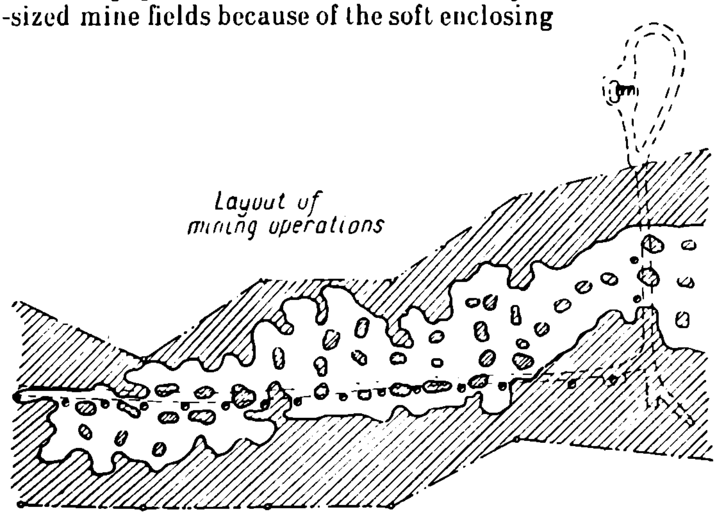



*Fig. 50.* Opening up of a gently dipping copper ore deposit in Dzhezkazgan

rocks and shallow occurrence. The layout of these fields is shown in Fig. 51.

The size of mine fields, lengthwise the placer, depends on the depth at which mining is done and the type of underground haulage facilities and varies from 100-200 metres (when the placer is 10-20 metres from the ground surface) to 400-600 metres (when the depth is 30-40 and more metres). This size may also be estimated theoretically. The crosswise extent of the field is equal to the width of the placer. In very wide placers, two or three rows of mine fields can be arranged instead of one.

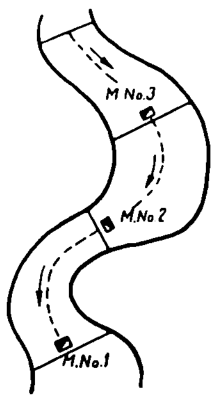


*Fig. 51.* Opening up of a placer

As a rule, mine fields are opened by vertical shafts. But it can also be done through inclined shafts and adits, this depending on the depth of the placer occurrence, surface topography and properties of rocks capping the metalliferous bed. The inclined shafts are particularly convenient when equipped with belt-conveyer and skip-hoisting facilities (Fig. 52). Adits are sometimes used to open *bench* placers lying above the level of existing rivers. If the valley with the placer slopes slightly and the metalliferous bed does not lie deep, the adit opening up the placer may be made to run along the thalweg of the valley, near the lower boundary of the placer. This opening is then excavated by a trencher as an open *ditch* for some distance from its mouth and is later timbered. Further on, the ditch runs into the main haulage drift extend-

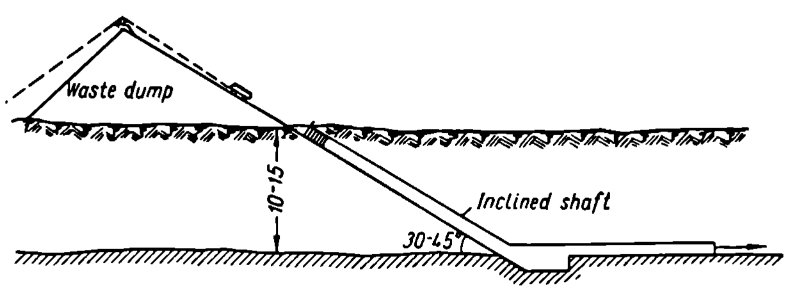


*Fig. 52.* Opening up of a placer through inclined shafts with skip hoists

ing along the valley thalweg. The trench is reinforced with timbering which is covered with moss and loose earth to protect the trench from precipitation.

Of especial importance in opening placer deposits is proper *drainage* of mine fields (see Chapter XX).

## 3. Opening of Inclined Ore Deposits

Tabular deposits, veins and other ore bodies dipping at angles between 30 and 45° can be opened through inclined shafts when they reach the surface (Fig. 53). There are a number of objections to driving shafts in the ore body proper: the occurrence of the ore body may be geologically impaired; its bottom may be very irregular; the method may involve large losses of mineral in shaft pillars. That is why the driving of inclined shafts "in ore" (dash lines in Fig. 53) is often rejected in favour of openings in the country rocks of the hanging wall, which are connected with the ore body by short crosscuts (continuous lines in Fig. 53). Such inclined shafts can be driven straight along the dip of the ore body as a whole, irrespective of the details of its structure or minor geological disturbances.

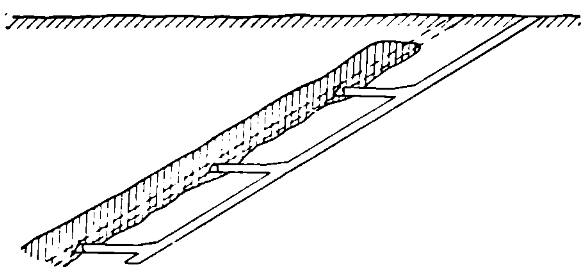


*Fig. 53.* Opening up of a dipping ore body through an inclined shaft

As mentioned above, many of the bauxite deposits occurring on the karsted limestone surface are distinguished by their very irregular bottom. That is why the inclined shafts opening mine fields of dipping bauxite deposits in the area of Krasnaya Shapochka (Northern Urals) have been driven partly in the country rocks of the foot wall and partly in the "swallow hole", or protrusive sections of the bauxite deposit.

In the foregoing text (Chapter II, Section 6) stress was laid on the important role played by conveyer sets in hoisting loads along the inclined shafts of coal mines. Below are some examples of the conveyer hoisting plants used in foreign metalliferous mines.

A hoisting plant of this type with a capacity of 7,000 tons of ore a day is employed at the French Homécourt iron mine.

At the American Pend Oreille lead-zink mine ore is hoisted from underground by two belt conveyers operating in series. Their length is 360 and 450 metres, angle of slope 10 and 17° respectively, and total hoisting capacity 350 tons per hour.

## 4. Opening of Steeply Dipping Ore Deposits

Since mineral veins and ore bodies of other shapes occur mostly at high angles of dip, this should be regarded as a major factor in opening ore deposits.

In discussing methods of opening ore deposits, one should also reckon with the fact that rocks enclosing useful minerals are often rather hard and even very hard, and that makes it necessary to pay particular attention to reducing the amount of work done in country rocks during the construction of mines and development of new levels and to take appropriate measures to speed up the driving of permanent mine workings. This is of special significance in opening steeply dipping deposits with the attendant large volume of development work in country rocks, and then new levels have to be developed rather frequently.

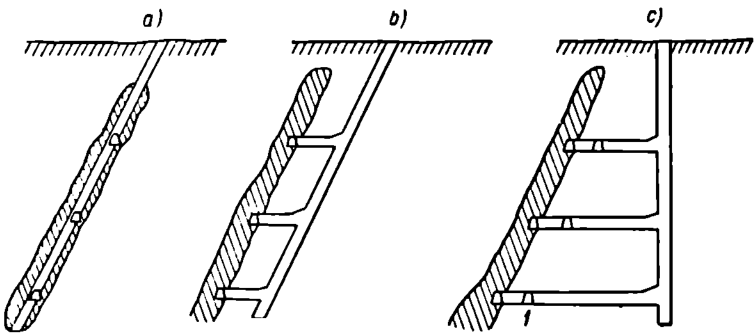



*Fig. 54.* Opening up of individual ore bodies

*a*—through an inclined shaft in ore; *b*—through an inclined shaft in the country rock of the foot wall; *c*—through a vertical shaft with crosscuts

Low-pitching ore-bearing veins can be opened through inclined shafts driven in ore (Fig. 54*a*), in the country rocks of a foot wall (Fig. 54*b*) and via vertical shafts with crosscuts (Fig. 54*c*).

The first method is applicable only when the position of the deposit is highly persistent and there are no geological distortions. It entails large losses of ore in shaft pillars. Hence, although the driving of an inclined shaft in ore means further exploration of the deposit and reduces outlays for its development, the method is not recommended.

The adoption of a vertical main shaft (see Fig. 54 *c*) has a number of substantial advantages (it facilitates sinking operations, hoisting of loads and men, etc.), but this method requires driving level crosscuts whose length increases with the decrease of the angle of dip of the vein, and it may prove unsuitable for the early development of a small lode deposit. Opening such deposits through an inclined shaft driven in country rocks and with high angles of dip (see Fig. 54 *b*) may prove economically justifiable and technically feasible, provided the shaft is equipped with a skip hoist.

When necessary in the early development of an ore body through inclined or vertical shafts, fringe drifts can be made, starting from crosscuts (*1* in Fig. 54 *c*). This method, however, may prove economically expedient only if the mineral reserves available in the level are not too small.

For reasons given above (Chapter II, Section 8), in mining series of ore bodies vertical and inclined shafts should be sunk on their foot walls.

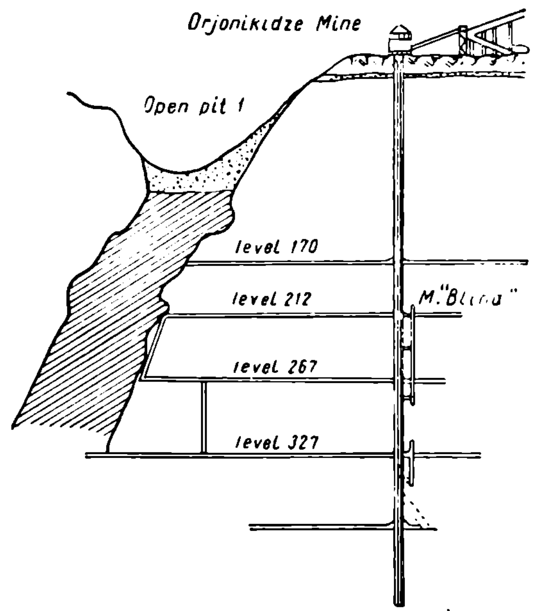



*Fig. 55.* Diagram illustrating early development of iron ore deposits at Krivoi Rog

In the opening of ore deposits, particularly those of small size, ventilating shafts are often located on the flanks of the mine field (and not next to the main hoisting shaft). This flanking location of ventilating shafts is convenient because it allows the use of the boundary ventilation scheme with unidirectional flow of the main air currents. This is a much desirable feature facilitating the removal of noxious gases following blasting operations, which are widely conducted in ore mining because of the strong ores and hard enclosing rocks usually encountered there.

*Series* of pitching lodes and other ore bodies are opened via vertical shafts and crosscuts. This pattern, for instance, is followed in the Krivoi Rog iron ore deposits (Fig. 55). The upper portions of thick deposits can be mined by the open-pit method.

When several ore bodies lie close to each other their combined development may be effected through fringe-level drifts (Fig. 56).

Blind ore bodies, which sometimes become apparent during the exploitation of a mine, are opened by various methods, depending

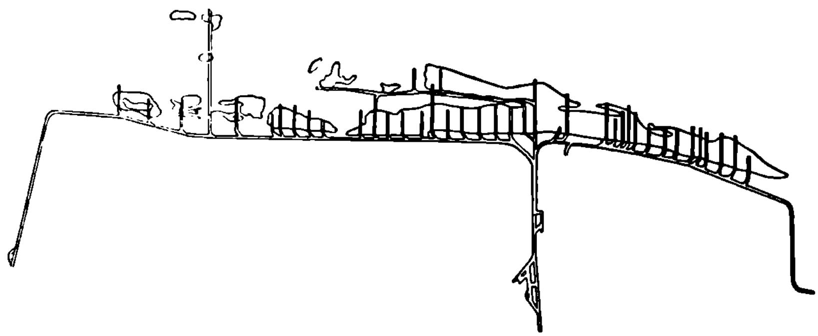


*Fig. 56.* Combined opening up of several ore bodies (plan)

on their position. Thus, Fig. 57 depicts a case when a vertical shaft and crosscuts were driven to open ore body *A*. Later blind ore body *B* was discovered at a greater depth and somewhat away from main ore body *A* and explored. In order to obviate deepening the main shaft and making extensive crosscuts, the blind ore body was opened through a separate blind shaft and crosscuts (dash lines in Fig. 57).

Subsequent development of blind ore bodies may entail additional outlays and operational expenses which can be avoided if the presence of a blind ore body in the deposit is known when the layout of the mine and its operations are planned. Therefore, construction of

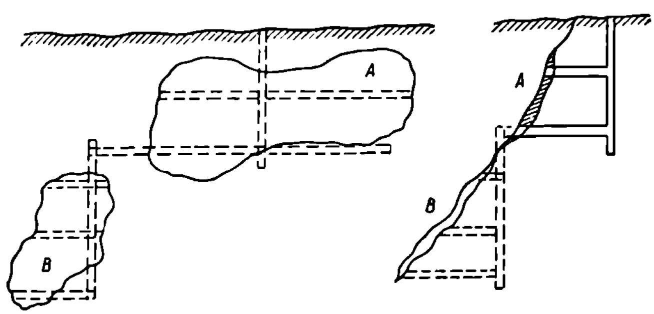



*Fig. 57.* An example of opening up a blind ore body

mines, particularly big ones, should be preceded by detailed prospecting and thorough exploration enabling timely discovery of such blind ore bodies.

Opening of a deposit by two methods—through a vertical shaft sunk from the surface and via a blind shaft—as shown in Fig. 57, is typical of a *combined* mode of opening.

Another example of this nature is the method of early development employed at one of the deepest South-African gold mines (Village mine, over 2,700 metres deep). Here the sheetlike deposit dipping at an angle of around 33° to the depth of 1,480 metres is opened through vertical shafts, and further down by two double-stage permanent rope inclines, driven in the rock formations of the foot wall.

There are many possible variants of combined opening, depending on the geological structure of a deposit and surface topography. The best is chosen by the procedure described in Section 18 of Chapter II.

## 5. Opening of Large-Sized Ore Bodies

This section is intended to help the student to get an idea of opening extensive ore bodies, whose thickness, length and width are of the same magnitude. Fig. 58 illustrates the development of such a deposit. Vertical and inclined shafts are driven in the enclosing

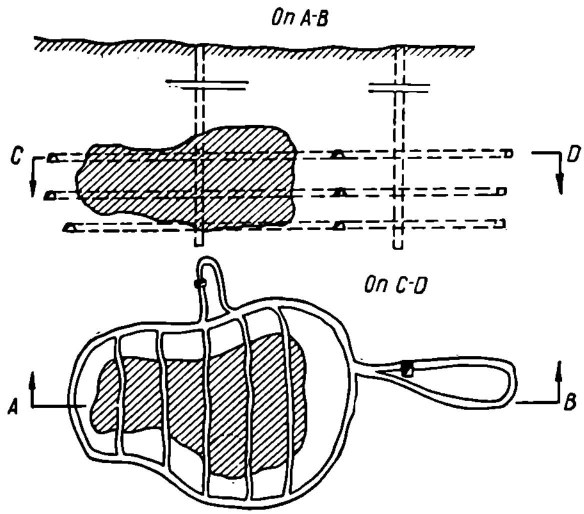



*Fig. 58.* Opening up of an extensive ore body

country rocks, outside the ore body, in order to avoid their being damaged by lateral shifts of the ground. Level crosscuts are extended from the shafts. On each level fringe drifts are made around the ore body. From these drifts development workings are driven within the levels themselves and the nature of these workings depends on mining methods (see Chapter XXI).

## 6. Level Interval Height in Opening Ore Deposits

In the development and mining of ore deposits, the vertical interval between levels is a factor of considerable importance, since in most instances ore bodies are opened with the aid of level crosscuts. The selection of the interval between levels is influenced by many factors: thickness of ore bodies, their structure, angle of dip, properties of the enclosing country rocks, method of mining actually adopted, cost of the development of new levels, etc. In many ore deposits these factors are by no means constant.

In selecting a level interval, one must take into account the combined effects produced by various factors, some of them of opposing nature. Because of advanced mechanisation there is a growing tendency towards increasing the formerly accepted level intervals.

In most cases, the vertical interval between levels of thick and relatively uniform ore bodies is set at about 60 metres and rarely at 80-100 and more metres. In the latter case, blind shafts equipped with lifts are excavated to facilitate the movement of men within the level and the transportation of machinery and timber. Low lodes and other ore bodies are usually developed by levels with intervals of about 40 metres, or around 60-70 metres if the level is divided into sublevels.

## 7. Annual Output and Service-Life of Metalliferous Mines

It is only in individual instances that ores extracted from mines can be shipped directly to smelting plants. Thus, for example, rich iron ores are used crude for blast-furnace smelting. Generally speaking, crude ores undergo *dressing* or *concentration*, *sorting*, *washing*, *mixing* with two or several other grades, and, finally, *average grading*, and, only thus *prepared for smelting*, are shipped to processing plants. Of particular importance for the metallurgical industry are the *concentrates* obtained through the benefication of crude ore. The quality of final concentrates depends largely on the useful components contained in the ore, the composition of its mineral fracture and the methods employed to enrich ore.

Iron ores require no concentration if their pure metallic iron content is not below 50 per cent and silica prevails in their mineral fracture, and not below 30-35 per cent with fusible mineral admixtures.

The metal content in manganese ores depends on their grade and ranges from 30 to 50 per cent. The copper content in copper ores usually amounts to 1 per cent, though sometimes it is considerably higher. The tungsten and molybdenum content in ores ranges from 0.1 to 0.01 and less than 1 per cent. The content of gold in ores extracted by underground methods is calculated in grammes or scores of grammes per ton of ore, etc.

Because of all this, the annual output of metalliferous mines is not expressed by the weight of the metal but by that of the ore, provided, naturally, it meets the quality specifications (*conditions*). The great variety of ore tonnage contained in deposits and the difference in the depth of their occurrence widely affect the capacity of metalliferous mines.

The biggest mines are those extracting iron and copper ores from rich deposits. Their annual output runs from a few hundred thousand to several million tons of ore a year and their service-life extends over decades. In some cases, mines engaged in winning silver-lead and certain types of rare-metal ores may also be considerable in size. The Canadian Froude-Scoby copper-nickel mine, for example, is equipped with a hoisting plant capable of delivering 14,000 tons of ore to the surface every day. The weight capacity of each of its skips is 13.5 tons and the plant is fully automated.

Mines excavating gold, silver, lead, zink, tungsten, molybdenum and some other ores produce from several scores of thousands of tons to several hundred thousand tons a year. In the same category are the capacities of mines engaged in working bauxite deposits. The service-life of metalliferous mines of small annual output may be reduced to 5-10 years.

In areas which have accumulated sufficient experience in working ore deposits, the annual output of a mine may be estimated by the simple though very imperfect method of using the so-called *coefficient* or *factor of utilisation* or *exploitation*. That usually means the average amount of ore obtained annually from one square metre of the ore bodies worked by the mine. This is a statistical value based on production figures recorded over a number of years. But with better managed operations in the mine or, what is more important, with new mechanical facilities or new mining methods, the working of deposits can perhaps be organised more effectively. Bearing this in mind, rough estimates of annual output by the coefficient of exploitation method should be made with great discretion and reservation. It is advisable to select the most suitable mining methods and accordingly set down the expected coefficient of exploitation.

Sometimes an analogous value, the average *annual advance in depth* of mining operations, is used instead of the coefficient of exploitation. When the total area of the ore bodies, the unit weight of the ore and the coefficient of recovery are known, this value helps easily to arrive at the ultimately annual output sought. For example, in the Krivoi Rog iron ore basin the average annual coefficient of advance in depth for mining operations in the next few years should be around 15-17 metres, while in the case of very thick ore bodies it reaches 10-12 metres and in that of thin deposits 25-30 metres.

At all events, the annual production capacity of a mine engaged in working ore deposits should be computed on the basis of time required for the extraction of the ore in one level and the reserves available in all the levels on the one hand, and of the time interval needed for the development of a new level, including time margin factor, on the other.

In the instance of ore deposits with strong wall rocks, the question of developing new levels is a crucial one, since the duration of development work should not exceed the time limit set for the extraction of ore reserves in the level, which, in turn, is closely bound with the amount of these reserves and the annual production capacity of the mine.

CHAPTER IV

# CHOICE OF SITE FOR SHAFTS

## 1. Factors Influencing the Location of Shafts

The location of shafts in a mine field is primarily determined by the method of its opening and development. The ultimate selection of the shaft location, however, implies additional consideration of the combined effects produced by many other factors.

The position of shafts in a mine field determines the aggregate extent of main mine workings, the cost of their maintenance, expenses on haulage of useful mineral, filling, waste, as well as transportation of men and ventilation costs.

The site of shafts should be chosen with a view to minimising the total outlays per ton of workable reserves in the mine field. On the other hand, the location of shafts meeting this requirement should be checked carefully against other factors capable of influencing its choice.

These factors may be classified as underground and surface.

The first include: a) geological and hydrogeological conditions governing the driving of shafts and other openings; b) conditions incident to the shifting and subsidence of rocks over mined-out areas; c) losses of valuable mineral in safety pillars; d) adequate mine drainage; e) suitable location of shaft stations.

The group of surface factors influencing the location of shafts comprises: a) ground surface topography—viewed from the standpoint of the advantages it offers to the construction of a railway line and other transportation facilities to the mine, and of the layout of its surface structures; b) building properties of the ground; c) location of permanent and temporary water basins; d) possible flooding of the mouths of the mine openings with consequent sporadic water flows; e) buildings available on the territory; f) possible snow and rock avalanches.

It is only but natural that, due to local conditions, these factors do not all apply to every individual case and that not all of them are of equal importance.

## 2. Conditions Underlying the Opening up of a Mine Field

The economic factors of opening up a mine field are easy to determine if it is nearly rectangular in shape and its mineral reserves are distributed more or less uniformly. As stated above (Chapter II), early development of such a field provides, wherever possible, for the location of the main (hoisting) shaft on the line dividing the mine field into two equally large wings or flanks (see Fig. 3).

We have already shown the advantage of this location as regards the underground haulage costs, transportation of men, maintenance of level entries and ventilation expenses. This conclusion concerning the site of the main shaft in relation to the mine field strike refers, naturally, both to the opening of individual seams and to that of coal measures, irrespective of the angle of dip of the country rock formations.

As for the position of the main shaft in the direction across the strike, there are many variants. Let us discuss some typical examples.

To open up a mine field through inclined shafts driven in a mineral bed (see Fig. 8), the position of the shaft mouths should be determined by the outcrop of the bed.

In the opening of an individual flat-lying or inclined seam via vertical shafts (see Fig. 12), the location of the main shaft depends on the adopted relative size of the mine field sections up and down the dip.

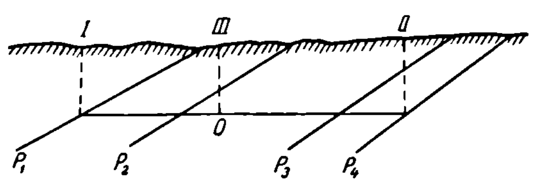



*Fig. 59.* Selection of the site of the main shaft in opening a series of seams through a single crosscut

Chapter II discussed the location of the shaft designed for developing a high-dipping seam (see Figs 18 and 19).

The problem is complicated by the opening of a series of seams. If, for example, a series of four seams $p_1$, $p_2$, $p_3$ and $p_4$ is developed through a permanent crosscut (Fig. 59), the depth of shafts in a flat country remains the same, irrespective of whether the main shaft is sunk at one of the ends of the future crosscut, in position *I* or *II*, or somewhere at point *III* between them. In such instances there may arise a question of placing the shaft at a point where the sum total of work done by the transport facilities along the crosscut, expressed in ton metres, would be at its lowest. This problem can be solved either graphically or analytically.

The following is the simplest way of solving it graphically. Let us assume that the mineral reserves in seams $p_1$, $p_2$, $p_3$ and $p_4$ are equal to $q_1$, $q_2$, $q_3$ and $q_4$ respectively and the distance between them to

$l_1$, $l_2$ and $l_3$. Let us mark off on straight line $AB$ (Fig. 60) the length of the crosscut of a certain scale and then plot on it the points indicating the position of loads to be trammed along the crosscut (that is, the intersection points of the seams and the crosscut where, presumably, entries are to be driven). Let us then assume that load $q_1$ is hauled in the crosscut to point *4*. The transport performance needed to accomplish this task will be

$$R_1 = q_1(l_1 + l_2 + l_3) \text{ ton metres.}$$

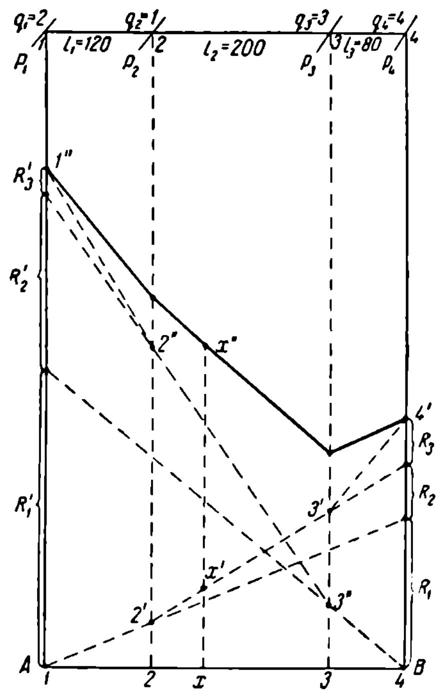



*Fig. 60.* Graphic solution of the problem of determining the optimal point of load concentration

Let us mark off this value of a certain scale on a vertical line traced through point *4*, and connect the upper end of this ordinate with point *1* by a straight line. The ordinates of this straight line will obviously represent on the same scale the performance needed to bring load $q_1$ to the point accepted as the starting point of this particular ordinate. Analogous lines should be plotted for other loads, assuming that they are hauled to point *4*. After that we plot a summarising broken line (line *1-2′-3′-4′* in Fig. 60). Ordinate $xx'$ of this broken line drawn through any point $x$ will represent the haulage performance in ton metres in the tramming to point $x$ of all the loads located to the left of this point. For instance, the length of ordinate $xx'$ represents on the scale accepted for plotting in Fig. 60 the number of ton metres for transporting loads $q_1$ and $q_2$ to point $x$. By the same method broken line *4-3″-2″-1″* is then plotted, its ordinates serving to measure the haulage performance in transporting loads from right to left. Finally, let us plot a general summarising broken line *1″-4′*. Its ordinates represent the number of ton metres needed to haul all the loads to a given point. For example, ordinate $xx''$ shows the ton metres required to transport all the loads to point $x$.

In Fig. 60 the plotting is done to comply with the following conditions.

| Workable reserves, in million tons | Distance between seams, in metres |
|---|---|
| $q_1=2$ | $l_1=120$ |
| $q_2=1$ | $l_2=200$ |
| $q_3=3$ | $l_3=80$ |
| $q_4=4$ | |

On accomplishing the graphic plottings given above we find that the minimal haulage performance (1,040 thousand ton kilometres) can be achieved when the shaft is located at the point of intersection of seam $p_3$ and the crosscut.

The above-described diagrammatic method of establishing the optimal point for the concentration of loads by following a certain trajectory, which involves minimum transport work (in terms of ton kilometres), is distinctly graphic, but requires much time for preliminary calculation and plotting. Therefore, it would be useful to discuss an analytical solution of the same problem, which helps to arrive at extremely simple criteria for establishing the optimal point of load concentration.

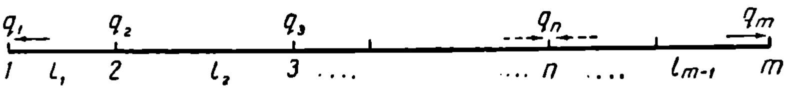



*Fig. 61.* A diagram for the determination of the optimal point of load concentration

Let us assume that loads $q_1$, $q_2$, $q_3 \ldots q_m$ are concentrated along a certain route (Fig. 61), at points *1*, *2*, *3*...*m*, situated at distances $l_1$, $l_2$, $l_3 \ldots l_{m-1}$, and that they should be hauled to one central point located on the same route. Let us find the optimal position *O* for this central point of load concentration to minimise the summary performance of the transport facilities (in terms of ton kilometres) in hauling all the loads to point *O*. The sum total of all the loads is denoted as *Q*.

Generally speaking, optimal point *O* may, depending on the absolute values and relative distribution of loads, be located:

1) at one of the terminals, that is, at points *1* or *m*;

2) at one of the intermediate points of load concentration (that is, points *2*, *3*...*m*—1);

3) somewhere in the sections between points *1*, *2*, *3*...*m*. Let us examine these three possible cases.

1. If the load concentration point lies at one of the terminals, for example *m*, loads can be hauled to it only from one side or direction (continuous arrows in Fig. 61). This point *O* may prove optimal for the concentration of loads only if the sum total of all loads delivered to it (that is, all the loads except the terminal one) is *smaller*

than the terminal load. In other words, $Q-q_m<q_m$, hence the condition for the coincidence of point $O$ and $m$: $q_m>\frac{Q}{2}$.

2. If point $O$ coincides with any of the intermediary ones, for instance, $n$, loads are hauled to it from two directions (dash lines in Fig. 61). Optimal point $O$ may coincide with the point $n$ only if the sum total of loads delivered to it from both directions is *smaller* than those which it would be necessary to transport from spot $n$ in a direction opposite to that indicated by dash arrows. If the sum total of the loads lying to the left of load $q_n$ is denoted by $\Sigma q_{left}$ and those situated to the right by $\Sigma q_{right}$, the condition cited above may be written down in the form of inequations

$$\left.\begin{array}{l}\Sigma q_{left}<q_n+\Sigma q_{right}\\ \Sigma q_{right}<q_n+\Sigma q_{left}\end{array}\right\} \qquad (1)$$

But since $q_n+\Sigma q_{right}=Q-\Sigma q_{left}$, the first inequation gives us

$$\Sigma q_{left}<Q-\Sigma q_{left},$$

hence

$$\Sigma q_{left}<\frac{Q}{2}.$$

Similarly, from the second inequation we arrive at the condition

$$\Sigma q_{right}<\frac{Q}{2}.$$

3. Should the load concentration point lie somewhere between points *1*, *2*, *3*..., the loads will have to be hauled to it from two directions, and it can obviously become optimal only if the sum totals of the loads delivered are equal. Otherwise, by shifting somewhat the load concentration point it would be possible to reduce the level of haulage operations. The condition providing for the balanced operation of transport facilities applies to the entire section of any given route.

The above-mentioned may be expressed by a simple rule: *optimal load concentration point O lies at a spot where the sum total of loads hauled to it from each direction is less than half of the aggregate loads; if in any section of the route the loads to be trammed come to half the sum of all the loads carried, point O may lie anywhere in this section of the route.*

This rule enables quickly to determine the position of the optimal load concentration point.

Thus, in the instance of the problem solved graphically in Fig. 60 we have found point *3* to be optimal. And indeed, since in this case $Q=2+1+3+4=10$, $\frac{Q}{2}=5$, the sum total of loads delivered to point *3* from left equals $2+1=3<5$, and from right the load is $4<5$, that is, the position of point *3* complies in every respect with the previously established analytical characteristics.

Worthy of note is the following: the position of optimal points is not affected by haulage distances, but depends only on relative tonnage and its distribution along the route.

If the opening is effected not by one but by several crosscuts, the selection of shaft location ensuring minimal haulage operations in

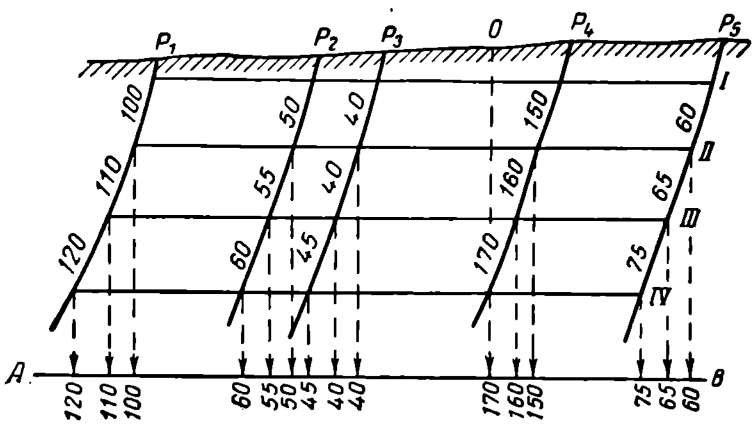



*Fig. 62.* Selection of shaft sites with minimal haulage along crosscuts in opening of a series of seams

all the crosscuts is done in the following manner (Fig. 62). In the detailed drawing of the opening up of the deposit we trace line *AB* parallel to the crosscuts, and plot on it the load concentration spots along the crosscuts. The actual values of tonnages to be hauled are marked at these points and after that the optimal point for the concentration of all tonnages is found on line *AB* by following the general rule.

The selection of the site for shafts offering the greatest advantages in opening is complicated by the irregular shape of the mine field or nonuniform distribution of its mineral reserves.

As an example let us consider a mine field in which the dip of the seam ranges from 15 to 58°, the field itself being limited by a fault lying at an oblique angle to the strike (Fig. 63). In a mine field like this, the reserves of the mineral per unit of length of the strike obviously will be greater in the flank with flat dip than in that with high dip. The sinking in this instance of the main shaft on line $xy$ which divides into equal parts the average *area* of the mine field along its strike would make it possible to reduce transport operations in the entries to the minimum. With such division of the shaft field into wings, the greater reserves of the flat flank and the smaller ones of the steep flank could be transported to the shaft over more or less equal distances. The shifting of the shaft location somewhat towards the side of the flat dip would reduce the total transport operations in the entries.

This problem can be solved correctly in the following way. As stated above, in the case now under discussion aggregate reserves

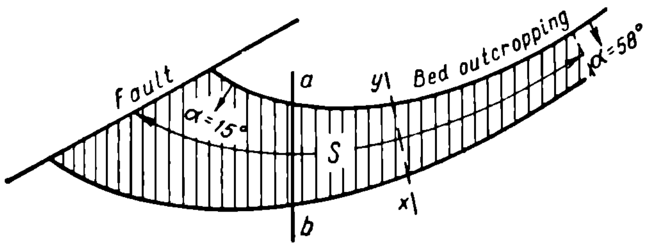



*Fig. 63.* A diagram for the selection of shaft location with nonuniform distribution of mineral reserves in the mine field

$Q$ of the useful mineral are distributed over the mine field continuously but not uniformly. Let us break these tonnages into elemental portions $\Delta q$, which we will regard as concentrated loads to be hauled in the entries. The smaller each element $\Delta q$ is, the closer the summary transport operations required to bring them to the shaft will be to the haulage performance of tramming the factual continuously distributed reserves of the field towards the shaft. By applying criterion (1) to the aggregate of such elemental loads, we find that within the limit inequations (1) become equation (2):

$$\Sigma q_{left} = \Sigma q_{right}. \tag{2}$$

In other words, with mineral reserves continuously distributed in the mine field, the optimal point of their concentration by haulage facilities lies on line $ab$, which divides the field *reserves* into two equal parts (and not its length along the strike). In a mine field of regular shape with uniform distribution of reserves line $xy$ dividing the length of the field and line $ab$ dividing its tonnage are coincidental.

It is not only the useful mineral that is hauled along crosscuts and entries in the mine field, but waste, and fillings too. These workings are also used for the movement of men and ventilating air currents. A problem analogous to the one of finding the best possible location of a shaft for transporting the mineral (a problem discussed above), may also be raised by each one of the cited factors taken individually. Moreover, if the amounts of filling materials or waste of the volume of ventilating air supplied to the stoping areas are proportional to the amount of the mineral mined, the optimal site of the shaft determined by each one of these factors usually coincides with that found in solving the problem of the most advantageous haulage of the valuable mineral. Quite often an analogous answer to this question is obtained in considering this issue from the viewpoint of the cost of maintaining entries.

It may generally be stated that the location of the main shaft, determined during the planning of the mine layout and envisaging minimum distances for hauling the mineral in underground workings, usually (but not always) also turns out to be optimal with regard to a series of other active factors, such as: waste tramming, maintenance and ventilation of mine workings, movement of men.

The configuration of shaft stations may also exert a certain influence on the final selection of the shaft site.

It is not infrequent that other considerations too prompt the introduction of certain changes in the position of the shaft, otherwise quite rational for the development of the mine (Sections 3-5).

## 3. Geological and Hydrogeological Conditions

The location of mine shafts must be chosen with due consideration of local geological and hydrogeological conditions in order thus to minimise the difficulties created during their sinking by the high water-bearing capacity and low stability of the rocks traversed. It is particularly essential to avoid intersection by shafts of *quicksands*, since this greatly complicates sinking operations. It often happens that the existing geological structure of the region and, particularly, the relief of bedrocks are such that the quicksands do not occur continuously but in "spots" or, at least, their thickness, the chief factor complicating sinking operations, is not the same over the entire mine field. Detailed borehole exploration of the section occupied by the running ground, its thickness and physical properties, helps to select the most suitable site for the shaft, one minimising the difficulties of sinking.

Although rigid rocks are a desirable factor in shaft sinking, it is nevertheless important to avoid traversing extremely hard rock formations, for example, quartzites. Driving in highly disturbed, faulted zones should also be avoided, for rock formations there may be not only badly crushed but also aquiferous.

A singular example of the influence exerted by the geological and hydrogeological structure of the deposit on the shaft location may be cited from the practice of the Kizel basin in the Urals. Here a coal-bearing formation of the carboniferous age has enormous series of limestone overlying and underlying it (Fig. 64). In many places these limestones are karsted. To a certain depth *ab* from the ground surface the karst cavities do not contain much water ("dry karsts"), but after that they are filled with it ("aquiferous" karsts). Driving shafts in aquiferous karsts is attended by considerable difficulties, as the experience of Kizel mines No. 1 and No. 6 shows. If sunk in position *I*, the shaft cuts across aquiferous karsts within section *cd*, and this is undesirable. Placed at point *II* the shaft does not traverse the limestones at all, but further down its location will require

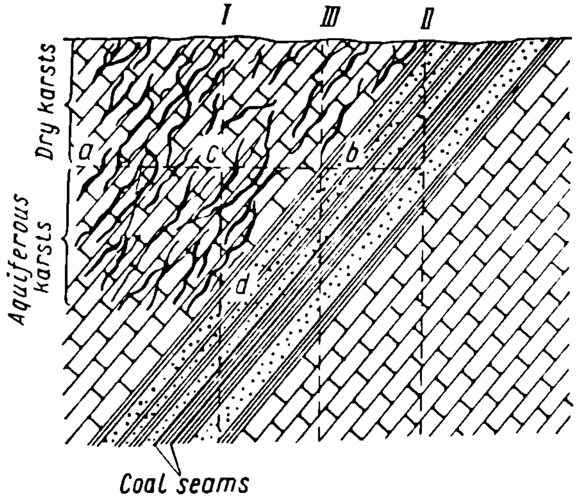



*Fig. 64.* Selection of the site of shafts in deposits containing aquiferous karsts

driving long crosscuts. Therefore, the most acceptable is position *III*, since in this case there is no need of traversing aquiferous karsts, and driving in dry karsts does not present any too great difficulties.

The Kizel coal fields are on the western slopes of the Urals. In the same latitude, but on the eastern slopes, lie extensive bauxite deposits, the mining of which is also complicated by the proximity of karsted aquiferous limestones to the ore bodies.

When country rocks enclosing any particular mineral deposit include limestone (or gypsum) series, mine planners should always reckon with the possibility of encountering aquiferous karsts, and this must be verified by preliminary hydrogeological exploration.

Not infrequently it so happens that the geological and hydrogeological structure of the deposits prevents finding the kind of site for shafts that would obviate the need of driving them in soft, running or highly aquiferous grounds. In such instances, depending on what the local conditions are, one resorts to some special method of shaft sinking (drilling by the drop-shaft method, cementation, freezing, pneumatic caisson, etc).

When deposits are nearly flat and enclosing rocks are highly watered (as is the case in most of the coal fields in the Moscow basin), the choice of shaft location is also influenced by the runoff and mine drainage conditions. When the main shaft is sunk to the lowest point of the bottom of the deposit, mine water can be made to run towards it by gravity. If the bottom is highly undulating, the runoff can be facilitated by auxiliary pumping stations. Because of that, when selecting shaft location in flat-dipping deposits, it is especially important to make use of mine plans with the floor structure contours plotted on them.

If the selectors of the site for the shaft should have any doubts about the adequacy and accuracy of information regarding the properties of the ground to be encountered in shaft sinking, they should make a test borehole at the place of the future shaft. This borehole should not be driven within the shaft outline, but at some distance away from it, so as to avoid the inrush of pressure water that might be in the rock series to the leading stopes of the shaft when it is sunk later on.

## 4. Subsidence and Movement of Rocks

The extraction of mineral deposits generally entails movement or shifting of ground over the worked-out areas. The possibility and expected nature of these phenomena must be reckoned with in each concrete instance (see Chapter XXIII) when selecting the site for the shaft.

When the shaft is to be sunk in an already partially exploited deposit, due notice should be taken of the relative position of the mined-out areas. Fig. 65 shows that seam $p_2$ has been worked out

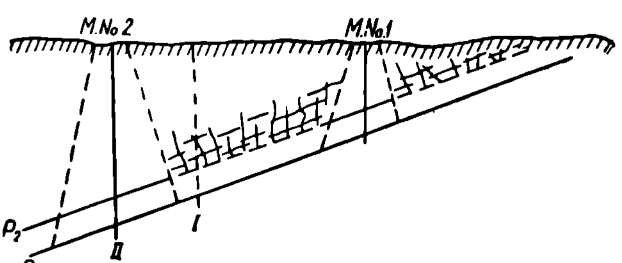



*Fig. 65.* Shifting shaft location to avoid traversing abandoned, mined-out areas

through shaft No. 1 over an area depicted by a dash line. Had permanent working shaft No. 2, sunk to mine seam $p_1$ at a lower level, been placed at point *I*, it would have crossed the old mined-out areas of seam $p_2$. There then might have been many difficulties created by possible accumulations of water, carbon dioxide or disturbed continuity of rock formations. What is more important, however, is that with the subsidence of the ground overlying mined-out areas still incomplete, the pieces of shaft lining would sustain considerable damage on account of the continued sinking of overhead rock formations. This could even lead to the deformation of the shaft axis, especially in steeply dipping deposits. Therefore, it is better to sink the shaft at point *II*, where it will traverse the intact portion of seam $p_2$.

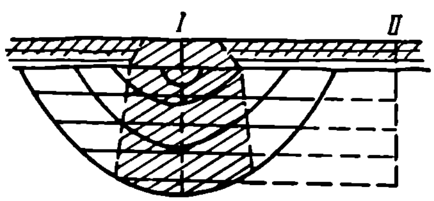


*Fig. 66.* Selection of shaft sites in the opening up of bed series occurring in a syncline

However, if the ground over the excavated area of seam $p_2$ has already subsided, the shaft may be sunk at point *I*. There have been cases of *yield* shaft timbering being used at the intersection of mined-out areas. Even the concrete lining of a round shaft may be made to yield, if it is done, for instance, on the same principle which underlies the design of the expansion pieces in mine-drainage pipelines.

To protect the shafts and surface structures of the mine from the damaging effects of ground movements (see Chapter XXIII) *safety pillars* with useful mineral are left near the former and under the latter. The choice of a suitable shaft location may either fully obviate the need of safety pillars or at least reduce their size to the minimum, and thus appreciably decrease the unwarranted losses of the mineral. Fig. 19, for example, illustrates that protective pillars in a thick bed may be avoided completely by shifting the location of the main shaft to the foot wall. Another typical example is given in Fig. 66, demonstrating the development of a deposit occurring in the shape of a syncline. Position *I* of the permanent working shaft reduces the cost of underground haulage, ventilation and aggregate crosscut driving to the minimum. But in these conditions, such a position of the shaft would necessitate leaving large mineral reserves in the safety pillars (shaded part in Fig. 66), a thing that could be dispensed with if the shaft were sunk at point *II* outside the bed series.

## 5. Surface Topography

In the case of minerals extracted in large amounts (mineral coal, rock and potassium salts, iron ore, some ores of nonferrous metals, etc.) it is highly desirable to have a full-gauge railway siding leading directly to the mine. This is not only important for facilitating large shipments of coal or ores to the consumer, but also for the delivery of various kinds of supplies—timber, metal, machinery and other equipment—to the mine. It is of no less importance during the construction of the mine. Therefore, no effort should be spared to choose the site for the shaft and its surface plants which will make them accessible to a full-gauge railway side line.

The surface structures of a big modern mine are very large (see Chapter V). The site on which they are built should be sufficiently extensive. The location of the site in relation to the topography of the surrounding country should be such as to reduce to the minimum the volume of grading and earthwork. Inasmuch as the buildings and structures (headframes, trestles) of big mines are of considerable dimensions and weight, the selection of the construction site should be preceded by the study of the ground to decide the size and design of substructures.

There are instances when, because of the local topographic conditions, it is impossible to build a full-gauge railway side line to the mine. In that case, narrow-gauge railway tracks, aerial tramways or conveyer lines must be arranged. At small mines, highways should be built for trucks. In this instance, of course, the location of the shaft with the adopted type of surface transportation should be so selected as to minimise both the first and the operational costs.

In mining deposits by filling the worked-out areas, one should take into account the convenience of delivering filling materials to the mine.

To forestall the flooding of underground workings by surface water, the mouths of the shafts should be so placed in relation to bodies of running water (streams, creeks, rivers), or stagnant water (lakes, ponds, swamps) as to preclude their inundation when the water rises and overflows the banks. In this connection, due account should be taken of possible inrushes of water not only from permanent water bodies, but also from temporary streams caused periodically by thaw or by downpours in usually dry gulches, ravines and lowlands.

In wooded areas the construction site should be cleared of trees to prevent fire hazards.

In mountainous country the surface structures of a mine should be built where there is no danger of rock bursts, landslides and snowdrifts.

## 6. Active Factors Involved in the Selection of Shaft Location

And so shaft location is influenced by manifold factors. In each individual case they must be thoroughly scrutinised, one after another, and their combined effect should be taken into account when finally deciding the proper site for the shaft. Some of these factors may be contradictory.

For example, in shifting the shaft from the position best suitable for underground haulage operations, we may at the same time gain by reducing shaft-sinking costs, if by doing so we avoid traversing running ground, etc. Therefore, the main thing to consider in solving the complex problem of selecting the location for shafts is the maximum economy and the influence of all available factors. Whenever several variants are possible, they are discussed and compared both from the technical and the economical angles before the final decision is taken.

The foregoing text dealt with the selection of the site for the main shaft used for hoisting minerals. As for the auxiliary shafts (ventilating, supply and filling materials, etc.), their position is determined fully by the local conditions after the establishment of the main shaft location. For example, upcast shafts at big mines are usually equipped with ancillary hoisting plants and, therefore, are sunk in the vicinity of the permanent working shaft, naturally, with due consideration of the layout of the mine's surface structures and the shaft station workings.

## 7. Opening Through Adits

The method of opening through adits can be used in mountainous country.

An adit can be driven across the strike (Fig. 67), on the strike and in a diagonal direction, this depending on the position of the deposit in relation to the mountain slope surface.

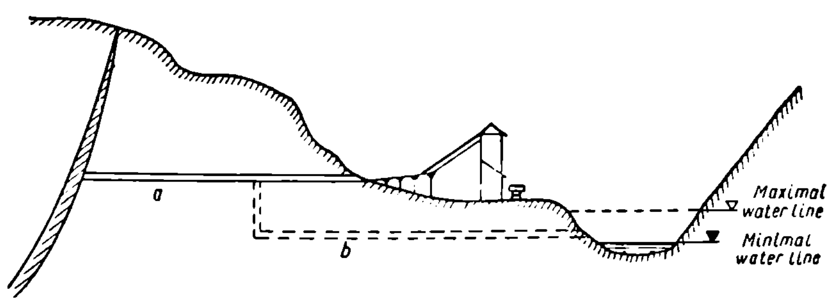



*Fig. 67.* Opening up of a deposit through an adit

Local topography and the position of the deposit permitting it, the adit may be arranged on a level that would allow us to work the greater part of the deposit above the level of the adit, that is, without having to hoist the mineral and pump mine water upward.

On the other hand, the mouth of the adit should lie above the level of the maximally possible rise of water in nearby basins. Spring high water or precipitation is not the same every year and, therefore, information on the maximal water table should be collected over as many years as possible. This is especially important in mountainous country, where small rivers in deep canyons or narrow valleys may, after abundant precipitation and intense thaw on high mountain peaks, swell very much and fast.

It is desirable to connect the mouths of adits designed for handling considerable tonnages with full- or narrow-gauge railway lines. In mountainous regions such lines are most conveniently built in the valleys (Fig. 67). A site of sufficient size should be made available at the mouth of the adit to accommodate technical plants and auxiliary buildings. If the nature of the deposit occurrence precludes immediate access of a railway siding to the mouth of the adit, highways for truck transport, aerial tramways or conveyer lines are built to link the adit with the mineral loading point at the trunk railway line or with the concentration plant. If the down-grade of a conveyer line exceeds 18-20° (Fig. 68), a *retarding* conveyer plant capable

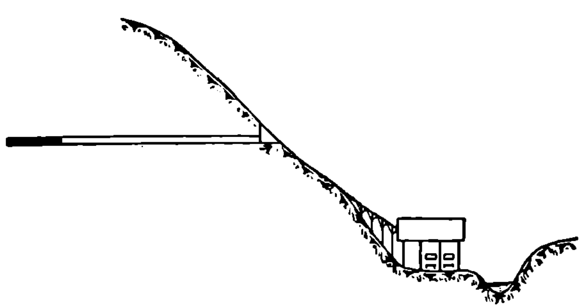


*Fig. 68.* Transporting the mineral from the mouth of an adit to the loading point by a conveyer line

of checking the sliding down of the material and handling it at a uniform speed is used. When the excavated mineral requires dressing, one must take into account the location of the concentration plant, dumps and loading points. To enable the mineral to pass through dressing units down the grade, without any intermediate lifts, the buildings of the concentration plant should be arranged on the slope of the mountain in the sequence required by the dressing operations (Fig. 69).

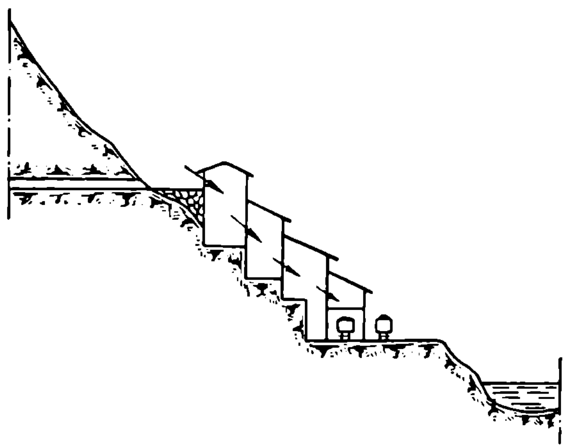


*Fig. 69.* Diagram showing the arrangement of concentration mills at the mouth of an adit

In selecting the site for an adit, it is also necessary to take into account the location of future dumps for the waste from underground workings or for mill tailings.

Being an opening with the mouth on the ground surface and having a certain gradient towards it to facilitate transportation of the mineral, every adit may be used as a runoff for mine water and for this purpose has drain ditches arranged in it. But there are also instances when special *drainage* adits are driven. They were of particular importance for mining operations in the old days before the invention of pumping engines. But even in our day, when mine workings are arranged in a suitable way, such adits can be used to a great advantage in the disposal of water by gravity. The mouth of special drainage adit *b* may be located considerably below the elevation of main, haulage adit *a* (Fig. 70). One of the major advantages of drainage by gravity is its dependability in the event of suddenly increasing inflows of water in the mine, since the capacity of

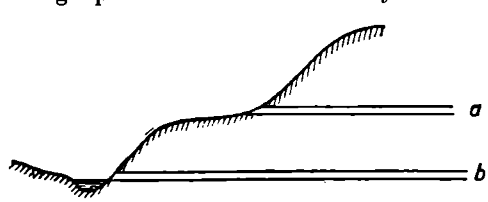



*Fig. 70.* Sketch illustrating the relative position of haulage and drainage adits

drain ditches can be raised substantially. In rigorous climate, the water passing down the mine ditches may freeze, this diminishing their useful section and causing water to overflow the floor of the adit and congeal. To avoid this, the ditch is dug deep over a distance of 100-200 metres from the adit mouth and is, moreover, heat-sealed in winter.

It is better, though much costlier, to drive a drainage adit over the same distance (see *b* in Fig. 70). To prevent the entry of too strong a current of cold air in winter months, ventilation doors are arranged at the adit mouth. The gradient of 0.004-0.008 towards the mouth of haulage adits is more than sufficient for the mine water runoff. Special drainage adits may have a slope of 0.001-0.002 and even as little as 0.0005, provided they have a large useful section for the flow and the water is clean.

Inasmuch as adits are arranged in mountainous regions, it is imperative to pay particular attention to possible sloping landslides, rock falls, mudflows, snow slides and avalanches. One should bear in mind that in areas exposed to the hazards of earthquakes, the cited phenomena may occur when the ground surface slopes less than usual.

In favourable natural conditions, opening through adits is a technically feasible and economically profitable method. Adits are particularly suitable for the combination of mountainous relief of the surface and flat-dipping deposits. One of the factors making for high labour efficiency in the U.S. coal industry is the availability of these conditions, which permit the use of adits in developing the greater part of the mine fields (about 60 per cent) in the country.

CHAPTER V

# SURFACE PLANTS AND STRUCTURES OF A MINE

## 1. Surface Structures at a Mine

The technical, administrative and auxiliary plants and buildings on the surface of modern big mines are designed to serve manifold purposes and, with the mechanisation of mining operations, are distinguished by the complexity of their arrangement and equipment.

Directly over shafts are the *headframes* of the hoisting plant and *shaft* or *head houses*, equipped for man hoisting, delivery of mineral and waste from underground workings, lowering of timber and other supplies, as well as pieces of machinery, etc. A portion of the shaft house or a separate building is used to accommodate *hoist engines*.

The arrangement of shaft houses and landing of the mineral and waste taken out of the mine is greatly simplified by the use of skips and tilting cages for load hoisting, and of conveyer lines in inclined shafts. Rapid and mechanised loading and unloading of mine cars is ensured by the employment of electric or air-operated pushers in cage hoisting.

As a rule, the mineral hoisted from the mine is hauled by mine cars or conveyers to the *receiving bins* set up alongside the railway tracks. In order to ensure continuous operation of the mine, which may be disrupted by delays in the shipment of mine output directly from the bins, *mechanised dumps* for the mineral are arranged at the mines. At coal mines they are provided with slushers.

If and when necessary, prior to its shipment to the consumer, the useful mineral undergoes sizing at *sorting plants*, or is sent to *concentration mills* to have the gangue and impurities removed.

Waste hoisted from the mine is transported to *waste piles* by narrow-gauge mine car, big dump-truck, conveyer or aerial tramway. In a flat country these piles are usually conical in shape (stocking by truss trimmers). These same dumps receive tailings from concentration mills and ashes and slags from boiler plants.

But if the mine area is crossed by ravines and valleys, the waste taken out of the mine should be used for filling them. When this is done, they should be levelled out and planted up. The gangue from waste piles may also be employed as a mine-fill or building material. If the waste contains carbonaceous substances, they should be given time to "burn out" before it is used as a mine-fill.

In any case, waste dumps should be far away from the mine yard and residential quarters because the carbonaceous matter emits noxious gases when it is burned out.

A mine is aired with the aid of big, powerful *fans*; in winter it is kept warm by steam *air heaters*. By special permission from the competent authorities, fire heaters may be employed in exceptional cases.

*Power substations* are built on the surface to supply power for the underground plants and machinery and other purposes.

Mines employing pneumatic equipment require *compressor* plants with water-cooling towers and spraying ponds for cooling circulating water.

In order to supply hot water to air heaters, mine change- and bath-houses and to heat the buildings in winter, *boiler plants* are built, in which fuelling and removal of ashes and slag are fully mechanised.

Machines and other items of mine equipment are overhauled at the mine's *repair shops*. Spare parts and other supplies are kept in special storehouses.

Timber is stored and treated in *timber yards*. Mechanised framing of timber end-points is done by timber-framing machines.

The materials and supplies delivered to mines are loaded and unloaded with the aid of cranes, single-rail devices and travelling loaders.

Cranes have latterly been introduced at big mines to ensure rapid unloading of timber from railway cars in "packages", their load-lifting capacity being as high as 30 tons.

Adequate communications and transportation of loads at the mines are ensured by narrow-gauge railways, well-paved roads and pavements built between individual buildings and storehouses.

For administrative and sanitary-hygienic needs of the miners and office workers there are special combined *office-change-and-auxiliary-service houses*. These are architecturally well-designed buildings with all the necessary amenities: cloak-rooms for ordinary and working clothes, shower baths, lamp room, that is, the room where miners' storage-battery and oil lamps are issued, turned in, charged and repaired; first-aid station; water filters; offices, etc.

No effort should be spared in planning the layout of mines and building them to reduce to the minimum the labour force required to service the surface plants. This may be achieved by making their layout pattern as simple, purposeful and compact as possible, by the mechanisation and automation of industrial operations.

These aims, the achievement of which ensures high efficiency for the surface labour force, are all the more attainable because the operations at the surface plants of big mines are limited to handling large volumes of the mineral, waste, timber and other materials that always move in the same directions: in the instance of the mineral—from the shaft to the receiving bins; in that of waste—to the dumps. It is this concentrated and constantly unidirectional flow of loads that offers the greatest possibilities for effectively utilising mechanisation, automation, telemechanics and remote control.

For one thing, this enables automatic control of load hoistings and of skip loading and unloading. Complete mechanisation and automation should be introduced in controlling the operation of landing chairs and handling mine cars in cage hoisting. It is quite feasible to record automatically mineral tonnages and volumes of waste hoisted from the mine. Centralised, or automatic, control may also be introduced for conveyer lines, feeders, loading and charging devices.

In timber yards mechanisation should be introduced in unloading timber from railway cars, its transportation over the territory of the yard, stacking and loading onto mine cars and trucks.

All this may contribute substantially to minimising the number of workers engaged on the surface and thus reducing their duties to controlling machines and industrial processes.

In planning and constructing new mines, the tendency should be towards making the surface structures and installations into a single architectural ensemble. All the roads and squares on the mine territory should be paved with asphalt and well-lighted, the whole area planted with trees and shrubs.

## 2. Arrangement of Mine Surface Structures

The arrangement of all mine plants, buildings and communications is determined primarily by their technological interdependence and topography of the surface. All these structures should be located in such a fashion as to minimise the volume of earth work. The orientation of the hoisting plant axes, the position of hoisting equipment in the shaft and the direction of the main shaft station workings should be well coordinated.

Let us discuss some typical examples of the layout of the basic surface plants and structures at the existing coal mines.

Fig. 71 illustrates a mine engaged in working a lignite deposit through an *inclined* main shaft equipped with a conveyer line and through an auxiliary vertical shaft with cage hoisting. It shows the following buildings and structures: inclined shaft *1* provided with a belt conveyer; covered conveyer gallery *2*; coal picking and sorting

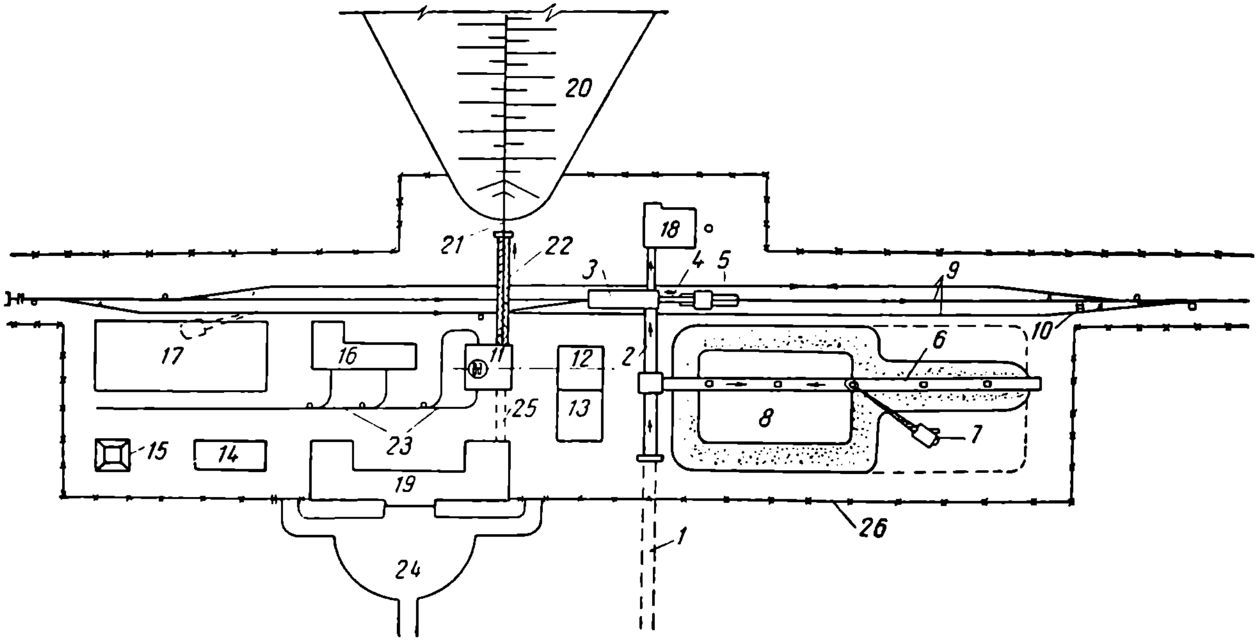



*Fig. 71.* Layout of surface structures in a lignite mine worked through an inclined main shaft

site with receiving bin *3*; loading conveyer *4*; railway wagon balance *5*; gallery for a reversible conveyer *6* for the removal and delivery of coal to the storage site; clamshell *7* for coal-loading operations at the storage site; coal pile *8* on the territory of the storage site; railway tracks *9*; trip-spotting hoist to move round railway cars *10*; shaft house of the auxiliary vertical shaft equipped with cage hoisting *11*; hoist house *12*; power substation *13*; compressor house *14*; water-cooling tower *15*; repair shops and storage house *16*; timber yard *17*; boiler house *18*; combined office-change-and-auxiliary-services house *19*; dump for the disposal of mine waste *20*; aerial tramway *21* to transport waste to the dump in special cars; protective wire net *22* under the aerial tramway and over the railway lines; narrow-gauge railway tracks *23*; highway *24*; traffic tunnel *25* for the passage of workers from the change house to the shaft house; fence *26* confining the mine yard.

Fig. 72 is illustrative of a surface structure layout at a coal mine with *vertical* shafts shipping run-of-mine coal. The scheme shows: shaft house *1* of the main shaft equipped with a tilting-deck cage hoist plant; hoist house *2*; receiving or shaft coal bin *3*; railway wagon balance *4*; conveyer gallery *5* for the delivery of coal to the storage place and boiler house; conveyer *6* to take coal from the storage site; emergency scraper coal storage site *7*; scraper hoist *8*; tail bogie of scraper hoist *9*; railway tracks *10*; trip-spotting hoist to move round railway cars *11*; air-heater unit to heat the intake air in winter *12*; power plant *13*; compressor house *14*; water-cooling tower *15*; repair shops and their storage-rooms *16*; timber yard *17*; boiler house *18*; combined office-change-and-auxiliary-services house *19*; waste dumps *20*; tunnel *21* for skip transportation of the waste taken out of the mine to the dumps; skip hoist plant for the disposal of waste to the dumps *22*; truss structure *23* for unloading waste-carrying skips; traffic tunnel *24* for the passage of workers; narrow-gauge tracks *25*; highway *26*; fence *27*.

Figs 71 and 72 depict the layout of surface plants and structures common to big mines now in operation. One essential disadvantage of these layout patterns is the high number of individual buildings and structures and their dispersal all over the territory.

That is why of late there has emerged a tendency towards integrating the plants and structures on the mine's surface into a very limited number of *building blocks*. Fig. 73, for instance, represents the layout of a standard mine with an annual capacity of 900,000 tons, designed by the Dniepropetrovsk Institute of Mine Designing, with only three such blocks. The first includes all the buildings and structures situated near the main shaft; the second those in the vicinity of the auxiliary shaft; the third the combined office-change-

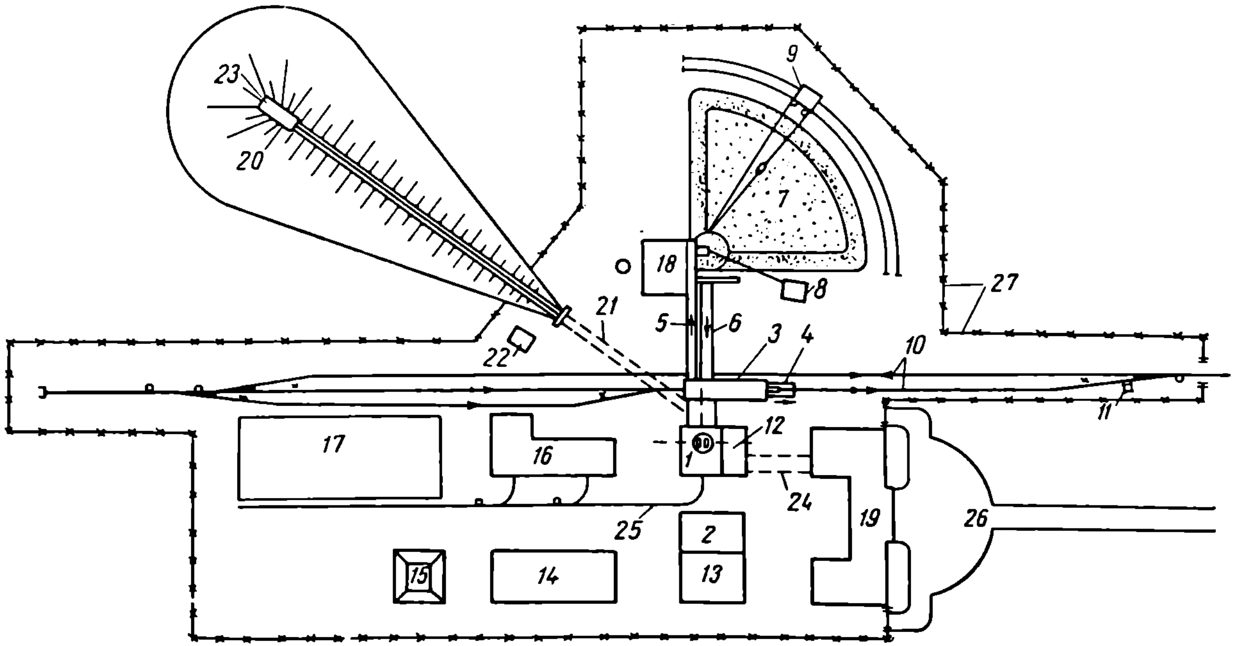



*Fig. 72.* Layout of underground structures in a mine producing run-of-mine coal

and-auxiliary-services house. The purposes the individual parts of the buildings serve are given in the caption to Fig. 73.

Such integration of buildings in separate blocks reduces the extent of tracks and roads on the surface of a mine and the water and heating, cable and sewerage networks, and at the same time makes it possible to decrease the total area of construction, which in the case of mines, apart from other advantages, also in most instances diminishes the size of safety pillars containing useful mineral.

Moreover, the small number of building blocks allows an effective employment of industrial methods of construction, that is, to use standard prefabricated details and structures, power cranes, etc.

Local conditions permitting, mine surface arrangement is simplified by designing centralised concentration mills, timber yards, storehouses and repair shops for two or several mines at a time. This creates favourable conditions for adequate mechanisation and automation of pertinent industrial processes.

When coal from the mine is transported to a nearby central dressing mill, the mine may dispense with a storage site, thus greatly simplifying its surface layout. In this case, coal may be shipped to the dressing mill not only in return-bound railway cars, but also by conveyer lines and aerial tramways.

Figs 71-73 give merely a schematic layout of mine structures. This arrangement should be defined more precisely in conformity with the features peculiar to the construction site and, above all, to local topography.

The drawings show only industrial and certain auxiliary and administrative buildings and plants of the mine. The general mine plan also has miners' towns with all their community and recreational establishments. It also depicts communications, electric power transmission lines and water mains, powder houses situated in the area surrounding the mine and ancillary industrial enterprises, that is, building-material quarries, brick works, etc.

## 3. Arrangement of Shaft Houses

The most typical of mine surface installations is the *shaft house.*

When the mineral is hoisted in *skips* or *tilt-deck cages*, their arrangement is simple and compact (Fig. 74). Cage *1* is unloaded over receiving chute *2*. Through deflecting gate *3*, which is moved by compressed air cylinder *4*, the mineral from the cage is fed to transfer bin *5* and the gangue to waste bin *6*. The latter also receives waste coming from the dressing mill by conveyer line *7*. From bin *6* waste is transported to a dump in cars *8* of the aerial tramway. Feeder *9*

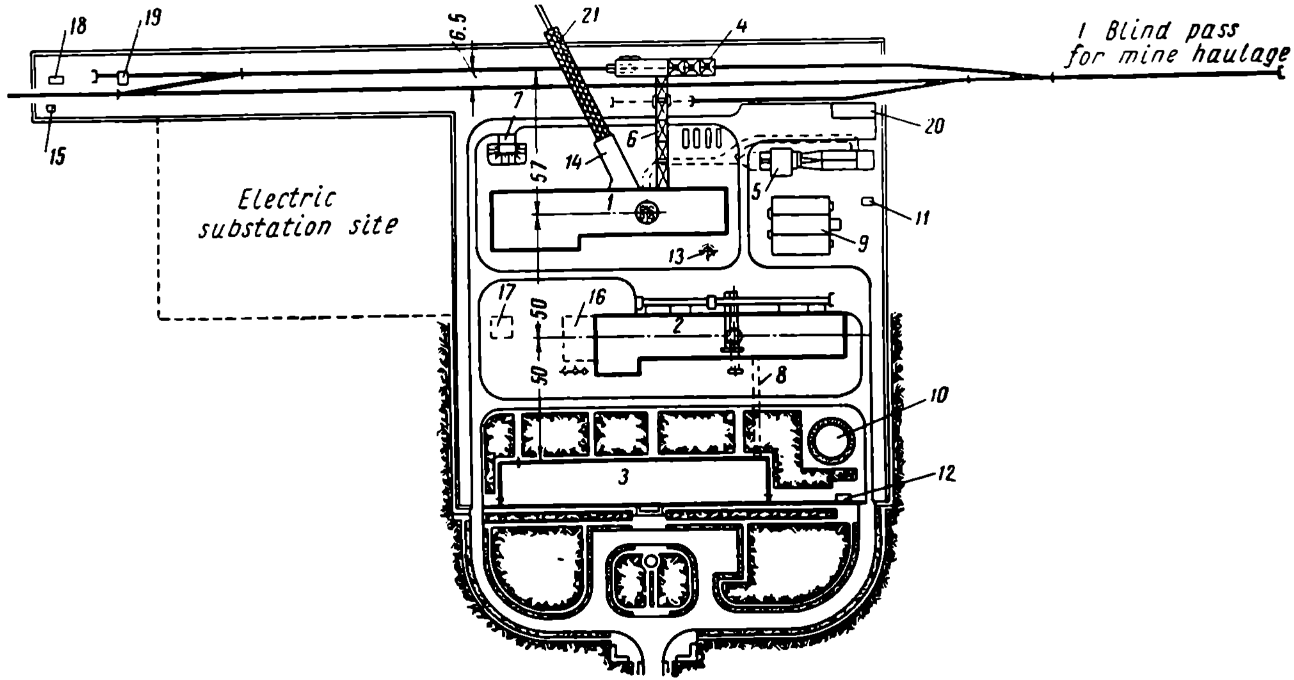



*Fig. 73.* General surface arrangement plan of a mine (basic variant)

*1*—main shaft block; *2*—auxiliary shaft block; *3*—office-change-and-auxiliary-services house block; *4*—receiving bins with a sorting plant and loading point; *5*—fan house; *6*—gallery from the main shaft block to receiving bins; *7*—storehouse for lubricating and lighting materials; *8*—men's passage tunnel; *9*—mine water settling reservoir; *10*—fire-fighting water storage reservoir; *11*—chlorination room; *12*—entrance-gate booth; *13*—stack funnel; *14*—loading section of the aerial tramway; *15*—switchman's cabin; *16*—compressor plant site; *17*—water cooling-tower site; *18*—trip-spotting hoist; *19*—trip-spotting carriage; *20*—unloading ramp; *21*—safety bridges of the aerial tramway is set up under bin *5* that feeds the mineral to the loading shaft bin by conveyer *10*. All these devices are so arranged that the mineral and waste can be fed and transferred automatically.

When the mineral or waste is hoisted in *ordinary cages*, the arrangement of the shaft house becomes more complex. In this instance there are three basic patterns used for the transportation or movement of loads in the shaft house: closed-circuit, stub or spur, and the one with the so-called cross, or transverse bogies.

In the case of the *circular* scheme (Fig. 75), the cars containing the mineral or waste hoisted from the mine move by the force of their own gravity along the sloping track towards coal tipper *3* or waste tipper *4*, where they are dumped. Before the tippers are catches *2*. Then, also by their own gravity, the mine cars move back to the shaft, thus completing their circular trip. To compensate for

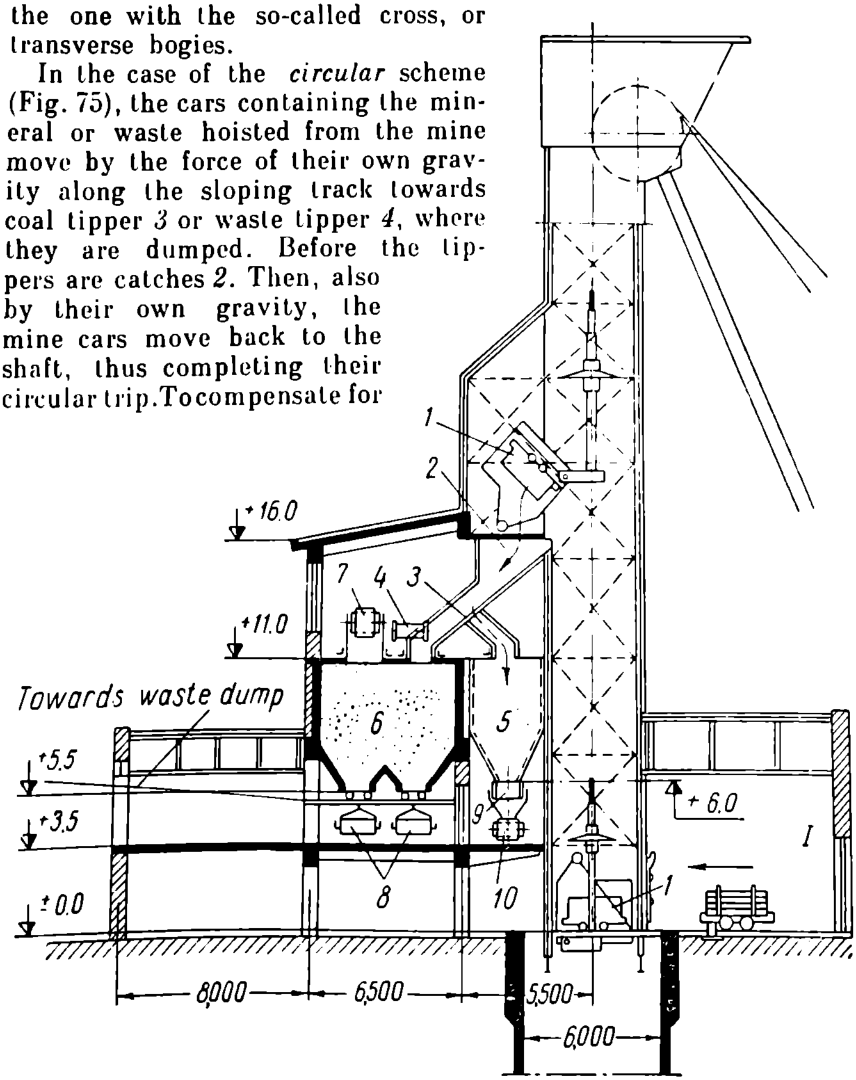



*Fig. 74.* Shaft house with tilting deck cage hoisting

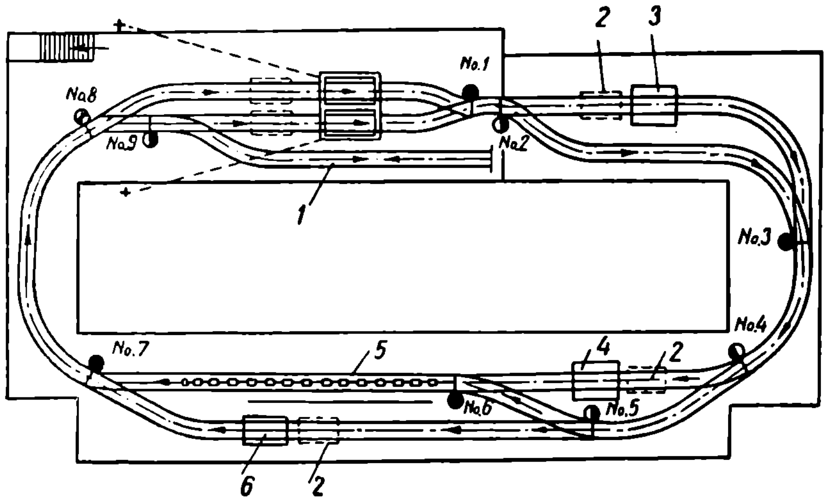



*Fig. 75.* Shaft house with closed circular mine car traffic

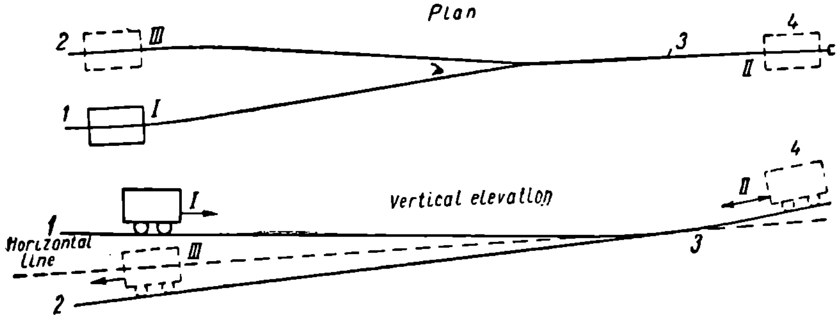



*Fig. 76.* Diagrammatic representation of a gravity yard

the elevation lost during their movement by gravity, the cars are lifted by incline hoist *5*, provided with an endless rope with catch cams gripping the car (*elevation compensator*). The cars with timber and other supplies are lifted to the shaft house by hoist *6*. A storage track for spare mine cars is depicted by *1*. The numbers in the drawing show the position of railway track switches.

To change the direction of haulage traffic *stub gravity yards* may be used instead of curves (Fig. 76). Cars are switched from gravity track *1* to gravity track *2* by connecting automatic switch *3*, behind which the tail track *4* is somewhat raised. The sequence of movement of cars is indicated by figures *I*, *II*, *III*. It is but natural that even when there are stub gravity yards the lost elevation has to be regained by means of incline chain hoists.

Shaft houses with circular or spur schemes of mine car traffic are rather cumbersome. Moreover, because of varying track resistance to the movement of individual cars, the latter, running along extensive gravity tracks, are either unduly accelerated or slowed down.

In view of this, the circular or spur traffic schemes are of late being replaced by *cross or transverse bogies* (platforms) (Fig. 77).

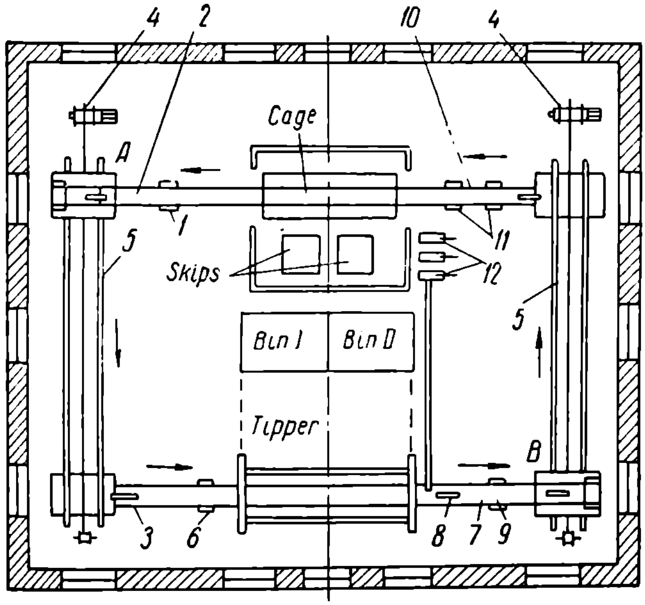



*Fig. 77.* Shaft house with transverse bogies

When the cage lands upon the chairs, its catches become automatically disengaged and the loaded car runs by gravity until it reaches catch *1*, which holds it as long as is necessary for bogie *A* to arrive. After that the catch automatically disengages, the car rolls along track *2* directly onto bogie *A*, and the latter is pulled up by hoist *4* along sloping track *5*. Thanks to this, the elevation lost during the movement by gravity is regained. Further on, the car rolls down into the tipper for unloading, whence, following track *7*, it mounts bogie *B* and then, by the uphill track moves up again to reach the cage via track *10* and go down into the mine. Car movements are controlled by catches *3*, *6*, *8*, *9* and *11* regulated by a topman with the aid of lever *12*.

Bringing mine waste to the surface in skips greatly simplifies the arrangement of the shaft house, while the flow of waste to the dumps is very easy to automatise. In view of this the plans for new big mines contain provisions for separate skip hoist plants for mine waste.

CHAPTER VI

# SHAFT STATIONS

## 1. Shaft Station Schemes

It has already been said above that a *shaft station* is an aggregate of underground mine workings located in the vicinity of the shaft and designed to serve the underground arrangements of the mine, as well as to connect the shaft with the main haulage and air openings.

The size and layout of shaft station workings differ very much, depending on mine output, the types of underground transport and shaft arrangements.

There are two types of shaft station intersections—*single* and *double*, this depending on whether the shaft has one or two outlets into the shaft station (Fig. 78).

The first type is, naturally, simpler and less costly, but its principal disadvantage lies in the fact that before a loaded car can be pushed into the hoisting cage the empty one has to be pulled out of it in the direction opposite to the first, and this requires considerable time and labour, since mechanisation of the process is rather difficult. With a double station mine cars are loaded and unloaded from the cage in one direction. This takes less time, the more so as this operation can easily be mechanised by the use of car pushers. Hence, single shaft stations are permitted, as an exception, only for prospecting or exploring shafts with low output and short service-life, or else for those with no hoisting plants, or with auxiliary plants operating irregularly.

To secure direct communication between the two sides of the station, a passageway for men is usually provided near the shaft under the ladder compartment of shaft *a* or *by-pass b* is driven, separated from the shaft by rock pillar *c* (Fig. 78).

The station is highest at the line of its intersection with the shaft—at least 4.5 metres with an arched ceiling and not less than 3.5 metres with a flat one, this being sufficient for long pieces, such as rails, pipes and timber, received from the shaft opening.

Hand-tramming of mine cars at the shaft station is permissible only in the instance of exploring shafts. Generally speaking, the transportation of the mineral and gangue at the shaft station should be *mechanised*. To accomplish this two principal methods are employed:

1) station workings are made either *level* or nearly so, while the loaded and empty trains are hauled by electric locomotives, some-

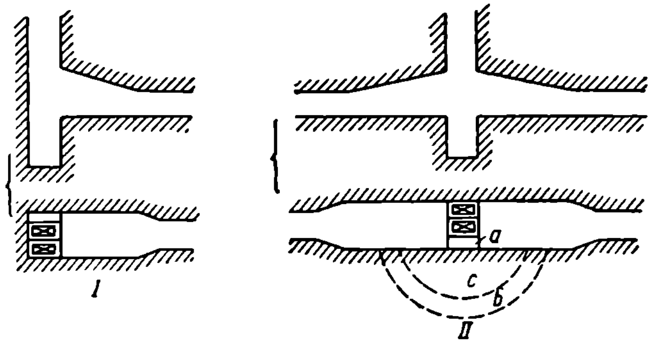



*Fig. 78.* Single (*I*) and double (*II*) intersection of a bottom station and the shaft

times aided by trip-spotting rope hoists. If the mine field has two flanks or a bilateral (in relation to the station) crosscut, the general pattern of train movements at the shaft station is as shown in Fig. 79;

2) station workings are made with a slope sufficient to enable automatic movement of mine cars by gravity, controlled by braking devices. The height loss is compensated for by inclined chain hoists, set up on the empties side of the station (*1-2* in Fig. 79) or by stump gravity yards.

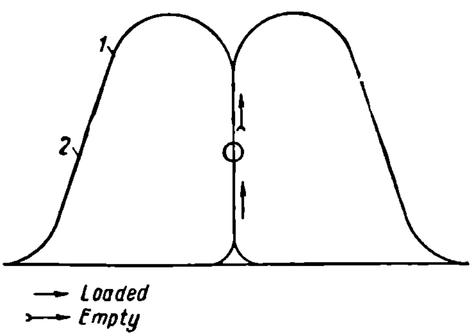



*Fig. 79.* Mine car traffic at a shaft station

In view of the predominance of electric haulage in the underground handling of minerals, preference is now given to the first scheme of car movements at the station, that is, to the use of electric locomotives and trip-spotting hoists and mechanical pushers in loading and unloading cages.

Standard layouts of shaft stations are shown in Fig. 80. These shaft stations are serviced by electric haulage with vertical shafts. In Fig. 80 the shaft is designated by a circle and the site for unloading mine cars with a skip hoist by a rectangle. The arrows indicate the main direction of movement. Layouts *1*, *2* and *3* refer to shaft stations where main hoisting is effected in cages. In the case of a *looping* layout, the station may lie perpendicularly (*1*) to the main haulageway or parallel to it (*2*). Fig. 80, *3* depicts a shaft station layout with a spur gravity yard.

Layouts *4*, *5* and *6* are designed for combined main skip and auxiliary hoisting in one shaft. Layouts *7* and *8* refer to skip and cage hoisting done through separate shafts.

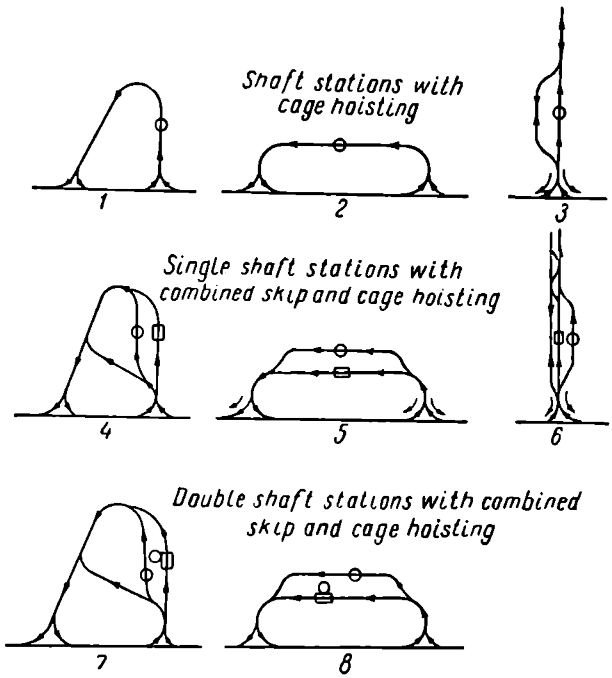



*Fig. 80.* Standard layouts of shaft stations

The extent of the loaded and empties track branches of the stations with main cage hoisting may be designed to accommodate one complete train each, while when main hoisting is done in skips the estimated length of tracks on either side of the shaft can hold 1-1.5 trains.

The layouts given in Fig. 80, although of standard type, do not cover all the possible cases that may arise in mining practice.

Of particularly simple arrangement are the stations of inclined shafts in which main hoisting is done by belt conveyers.

If some particular shaft is used for supplying filling material to the mine, the layout of its station should be designed so as to provide for the haulage of this material, etc.

All points considered, the layout of a shaft station should ensure the necessary traffic capacity, simplicity of switching operations, employment of minimum labour force, mechanisation of transport and its safety, and minimal volume of excavation work. To secure traffic safety and reduce the number of men working at shaft stations, wide use should be made of signalling and automatic devices.

## 2. Mine Car Switching Operations at Shaft Stations

Fig. 80 illustrates only the principal travel schemes accepted at shaft stations but subject to further elaboration. In Fig. 81, for example, we see diagram 7 of the Fig. 80 in more detail and on a larger scale, this enabling us to follow more closely the successive movement of mine cars.

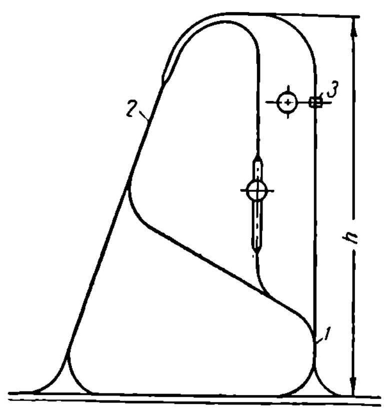


*Fig. 81.* Graphic estimation of the size of a shaft station

Loaded trains brought to the shaft station by electric locomotives pass by the curvature of the loaded track section and are then pushed backward into this section of the station. If there are only coal-laden cars in the train, the whole of it is pushed onto the skip tracks of the loaded section of the station, beyond the entrance switch and the switch at the branching out to cage track *1*. After that the electric locomotive is uncoupled and moves via a by-pass for locomotives (*1-2*) to the empties section of the station. If the train includes rock-laden cars, the electric locomotive will carry out the following switching operations: if the rock-laden cars are at the head of the train, it pushes the coal-laden cars onto the skip track branch (*1-3*) first and the rock-laden ones onto the cage track after that. If the rock-laden cars are at the tail-end of the train, the locomotive must first take them to the cage branch of the station and then the cars with coal to the skip tracks. This last operation is less convenient and, therefore, the cars with rock should be placed at the head of the train. Having delivered the cars with coal and rock to the loaded section of the shaft station, the locomotive proceeds via the by-pass to the empties track section. Here it is coupled to the empties train and moves to its point of destination. When the mine cars intended to make up the train are spotted on the skip and cage track sections

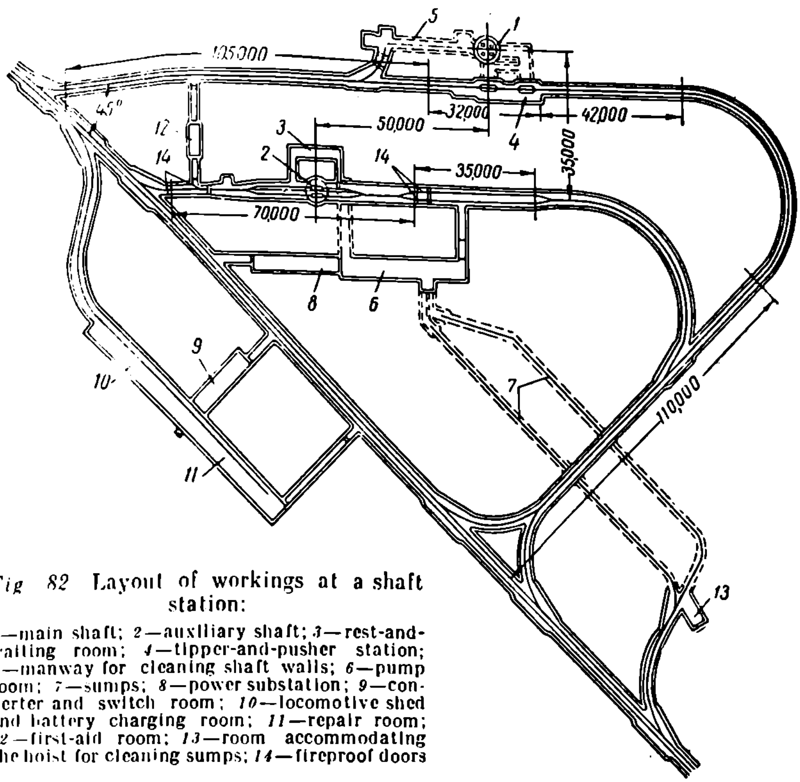



*Fig. 82.* Layout of workings at a shaft station:

*1*—main shaft; *2*—auxiliary shaft; *3*—rest-and-waiting room; *4*—tipper-and-pusher station; *5*—manway for cleaning shaft walls; *6*—pump room; *7*—sumps; *8*—power substation; *9*—converter and switch room; *10*—locomotive shed and battery charging room; *11*—repair room; *12*—first-aid room; *13*—room accommodating the hoist for cleaning sumps; *14*—fireproof doors

of the station, the locomotive gathers them first on one and then the other track.

The unloading of trains in tipper *3* is done *without uncoupling* the cars. Unloaded trains proceed along the empties way, where they are coupled to electric locomotives, and return to their sections.

The length of a shaft station ($h$ in Fig. 81) with a 900-millimetre track gauge is about 150 metres.

A shaft station of this type, with one train arriving on the average every six minutes, is capable of handling 360 tons an hour.

A more detailed layout of a shaft station at a big modern mine is given in Fig. 82, which shows not only mine tracks but also the location of service stations and the rooms with underground machinery and equipment.

An analogous layout of a station in an inclined shaft equipped with a skip hoist plant is shown in Fig. 83.

## 3. Service and Engine Rooms at Shaft Stations

1. *Shaft and skip measuring pockets.* In the case of *skip* hoisting, special capacities (*pockets*) have to be provided for the mineral loaded into the skips. These pockets have a capacity either equal to the payload of one skip (*low-capacity* pockets), Fig. 84, or to that of many skips (*high-capacity* pockets), Fig. 85.

Fig. 85 illustrates a loading device with sloping pockets, widely used in the iron ore mining industry. When unloaded, ore drops onto inclined bar grizzlies, where big chunks break up, while the fines pass freely through the bars. Under the grizzly, in the lower part of the funnel-shaped receiving chute is metal deflecting gate *1*. The pocket is divided by reinforced concrete partition wall *2* into two branches or compartments, each of which is designed to hold a certain grade of ore, or else one is used for ore storage and the other for barren rock. The inside dimension of the compartment is 1.7×3 metres. The gate is thrown over from one position to another by air motor *3*. A special manway is provided to service the deflecting gate. The pocket is concrete-lined. The lower inclined plane of the pocket, along which the ore slides down, is made either of concrete or reinforced concrete, but with a metal lining of railway rails, while the cheeks are faced with thick sheet steel. The angle of bottom slope

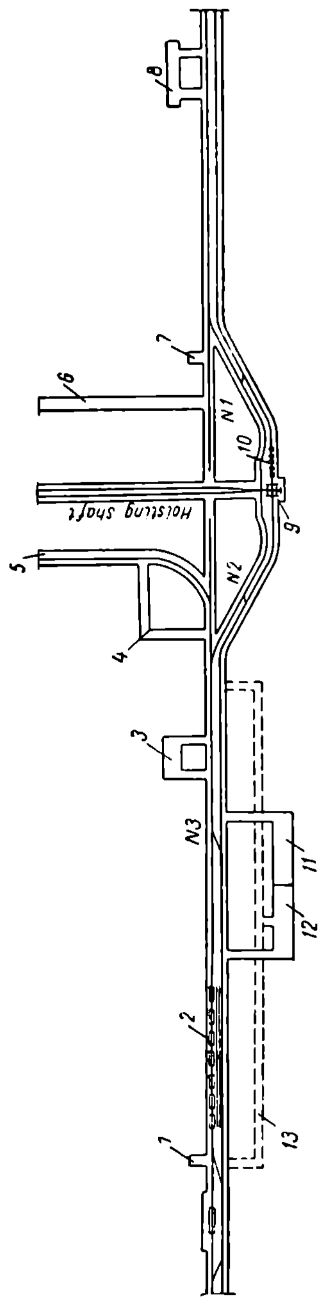



*Fig. 83.* Bottom station in a shaft with skip hoisting

*1*—storage room; *2*—locomotive shed; *3*—first-aid room; *4*—rest-and-waiting room; *5*—auxiliary shaft; *6*—air shaft; *7*—central underground control room; *8*—storeroom for fire-fighting materials; *9*—tipper; *10*—car pusher; *11*—electric substation; *12*—pump room; *13*—sump

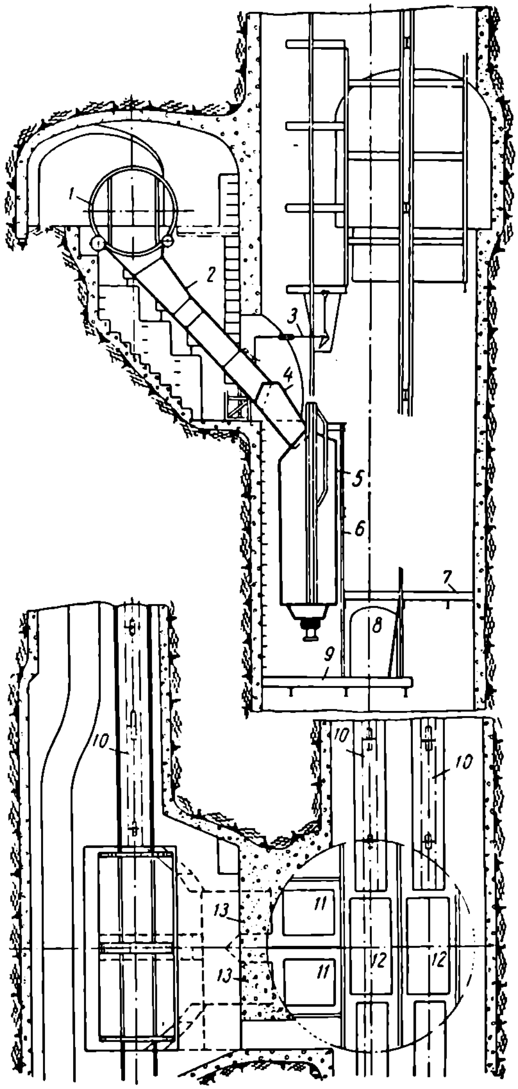



*Fig. 84.* Loading arrangement in skip hoisting with a low-capacity coal pocket

*1*—tipper; *2*—coal pocket; *3*—safety shutter; *4*—feed spout; *5*—skip; *6*—lining; *7*—safety platform; *8*—sump drainage pump room; *9*—protective cover (stage) against penetration of coal into the sump; *10*—car pusher; *11*—skip; *12*—cage; *13*—loading arms

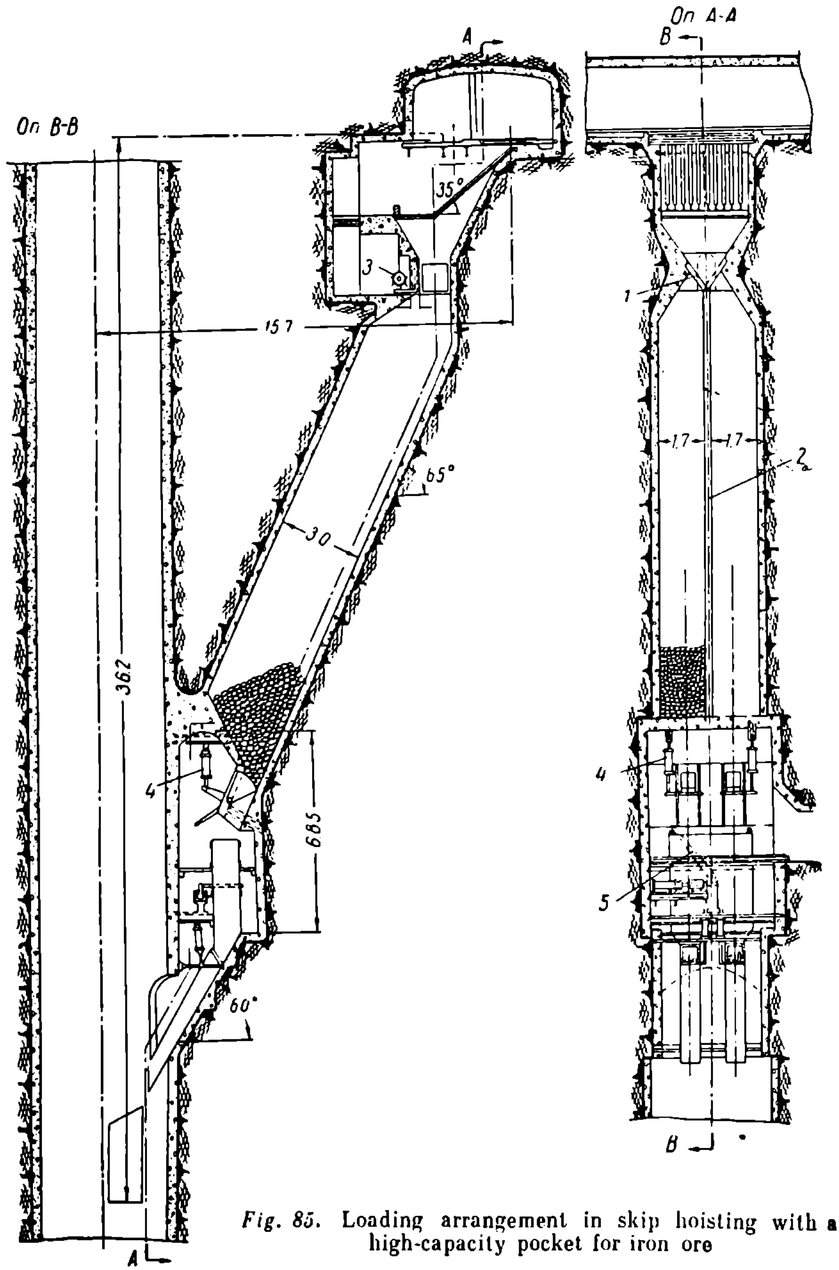



*Fig. 85.* Loading arrangement in skip hoisting with a high-capacity pocket for iron ore

in the pocket depends on the size of the rock and its hardness and humidity. With small-sized moist ore this angle comes to 76-75° and with large-sized dry ore to 55-60°.

The lower section of each compartment in the pocket tapers down somewhat and is furnished with an arc gate operated through air cylinder *4* hinged to a beam.

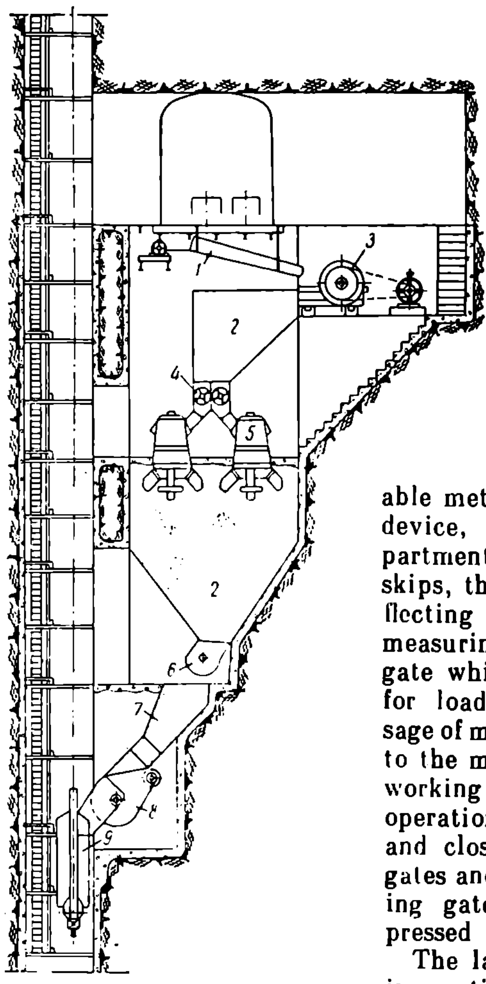



*Fig. 86.* Bottom station in a rock salt mine with skip hoisting and underground coarse breaking

*1*—flat grizzly; *2*—pocket; *3*—crusher; *4*—feeder; *5*—secondary breaker; *6*—bottom pocket gate; *7*—measuring bin; *8*—gate; *9*—skip

The inclined pocket directly abuts a *measuring bin* with a *measuring device* of rigid construction. The latter comprises measuring boxes or funnels that accurately measure out the volume of the mineral necessary to fill a skip. Hence the capacity of the measuring device is equal to that of the skip. The device is made of sheet steel 8-12 mm thick. On the inside it is faced with easily interchangeable metal. Thanks to the measuring device, ore from any pocket compartment can be loaded into different skips, this being accomplished by deflecting gate *5*. At the bottom of the measuring device is a cut-off or flat gate which opens in either direction for loading skips. To facilitate passage of men from the main station level to the measuring bin there is a special working with a ladder way. All the operations involving the opening and closing of pocket and measuring gates and throwing over of the deflecting gate are accomplished by compressed air cylinders.

The layout of main shaft stations is particularly extensive and complicated in cases when, apart from large pockets, there is crushing equipment underground to break the mineral, say, potassium or rock salts (Fig. 86). A vast shaft station with powerful breakers for underground crushing has been built in the U.S.S.R. at the potassium mine at Solikamsk.

In the U.S.A., thanks to effective mining machinery and high-capacity mine cars, underground crushing is practised very widely.

Out of the 69 iron ore mines in Lorraine, France, 20 were equipped with underground crushing plants as far back as 1952.

2. *Pump rooms and storage sumps.* Depending on local conditions, the layouts of storage sumps and pump rooms may be so different that it is practically impossible to reduce them to a few characteristic types. We shall, therefore, dwell on one such layout (Fig. 87) in order to elucidate the requirements which the underground pumping plants have to meet and which should be considered when planning the layout of shaft stations.

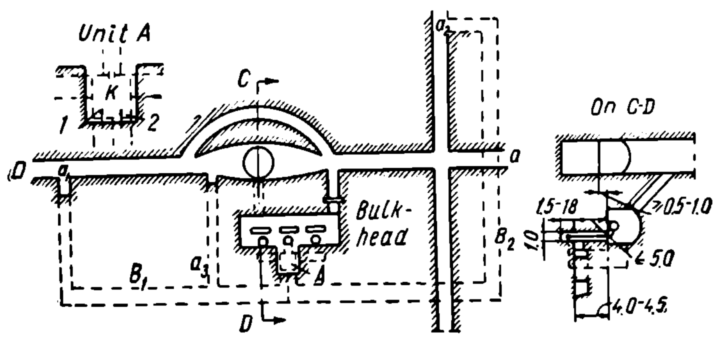



*Fig. 87.* Pump room and sump layout

The installations of the principal mine drainage plant are put up in the area of the main shaft station, since it is towards this point that mine workings normally slope and it is, therefore, there that mine water runs off from the whole or part of the mine field. Besides, since in most instances mine drainage pipes are laid in hoisting shafts, or in other mine openings located in their vicinity, the installation of pumping plants around the shaft station reduces the length of delivery pipelines to the minimum.

Underground drainage workings are made up of two principal parts: sumps and pump rooms or stations.

*Sumps* are the reservoirs whence mine water is disposed of by water-lifting machines installed in the pump room. At the same time sumps serve as *settling tanks*, where mine water loses some of the mineral particles it carries and enters the pumps cleaner. The sedimenting mineral particles gradually fill up the sumps, impairing their capacity and making it necessary to clean them periodically. To secure continuous operation of the pumping plant, the sump ordinarily consists of two separate compartments $B_1$ and $B_2$ (Fig. 87). Optimal settlement requires that water enter the sumps at points farthest away from the suction end. Thus, in Fig. 87, water from the crosscut enters sump $B_1$ at point $a_1$, and sump $B_2$ through $a_2$. In normal conditions both sumps operate simultaneously, but when one has to be cleaned it is shut off by a method described below. If it suits the location of mine workings, the separate compartments of the sumps may be of different capacity. In Fig. 87, for example, sump $B_1$ is smaller than $B_2$.

The overall size of sumps is determined not only by their live capacity, but also by the number of hours it takes normal inflow to fill them with the pumping plant shut down. The greater this time interval the longer the pumps can remain inactive. In certain respects this presents many advantages both when the pumping plants function normally and when one is forced to shut them down. According to standards, a pumping station is supposed to operate twenty hours a day even when the water inflow is normal. On the other hand, construction of extensive sumps requires appreciable initial outlays. By way of a compromise between these opposing factors the summary standard capacity of main sumps is now set at a figure equalling normal inflow of mine water for eight consecutive hours. In the Moscow coal basin this standard value has been reduced to four hours.

To avoid difficulties in driving and, especially, timbering of sumps, their cross-section is made to approximate that of ordinary haulage workings, that is, the larger their capacity the more considerable their length is. In order to cheapen them, they are driven in relatively soft rock but in the kind, of course, that would not require expensive timbering and frequent repairs. It is expedient, for instance, to make use of coal seams. There are many varieties of timbering one may use, this depending on the stability of rocks and the service-life of sumps. Most often it is the usual wood timbering.

The floor elevation of the sump in relation to the neighbouring workings is determined by the following considerations (Fig. 87).

The suction head is not to exceed 6.5 metres. The pump axis should be approximately 0.5-1 metre above the floor of the pump room. In turn, the floor of the pump room must lie 0.5 metre above the level marking the intersection of haulageways and the shaft station. Moreover, the inlet suction pipe of the pump is lowered somewhat below the bottom of the sump. By comparing these figures, we find that the floor level of the sump is around 5.5 metres below the floor of the pump room and 5 metres below the adjacent haulageways.

As we have already said, sumps also serve as *settling tanks* for mine water.

There is no set schedule for cleaning sumps—it all depends on the purity of the water entering them, but in favourable conditions it

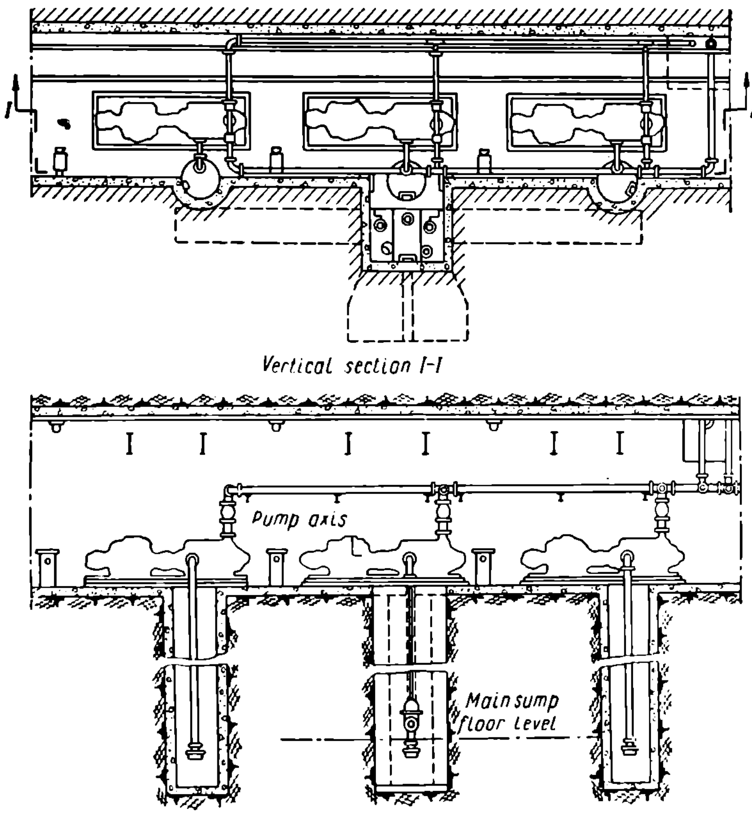



*Fig. 88.* Pump room

is done once or twice a year. To facilitate their cleaning, the sumps are made of two sections separately connected with drain or receiving pit $K$ located near the pump room. From the sump the water gets into the receiving pit only through a pipe furnished with a slide valve. Sump $B_1$, for example (see Fig. 87), is linked with pit $K$ by a pipe with slide valve *1*, while sump $B_2$ is connected with the same pit by a pipe with slide valve *2*.

The disposal of the dirt and mud accumulated in sumps, that is, their cleaning, is effected by a variety of methods. The work can be mechanised by employing mud pumps and pneumatic instruments.

In the main mine drainage system only electric centrifugal pumps are used. Normally, a pump station has three units—one in operation, the second in reserve and the third under repair. These three pumps are of equal rated capacity and designed to handle the daily inflow in twenty hours. To even up the wear of the pumps, each is operated ten hours daily.

During the spring floods and generally more intensive inflows two pump units may be operated simultaneously. If there are grounds to fear that the water flow may more than double, place should be left in the room for a fourth unit of the same size as the other three.

Normally, there should be three pump-discharge lines laid in the shaft. The diameter of each should conform to the output of one pumping unit and the water flow rate in the line should lie within the range of 1.5-2.2 m/sec.

The size of the pump room is determined by the overall dimensions of mechanical equipment (Fig. 88).

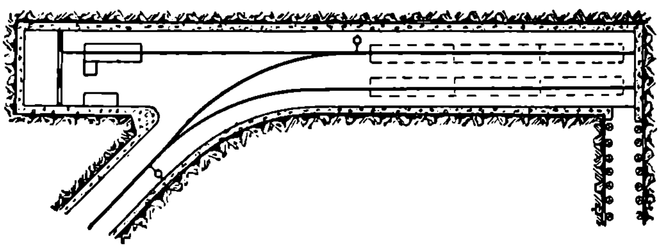


*Fig. 89.* Underground shed for trolley locomotives

To provide additional safeguards against flooding, the pump room can be sealed off in the following manner. Normally, the pump station is connected with other underground workings at three points: 1) with the sumps through the receiving pit; 2) with the shaft station through the manway; 3) with the shaft through the inclined passageway for the outlet of discharge pipes (see Fig. 87). The access of water into the receiving pit may be barred by appropriate check valves. The inclined pipeline way pierces the wall of the shaft at the height of 10 metres above the floor of the station. Consequently, the pump room can be isolated from water even when all the surrounding workings are flooded by a *watertight bulkhead* with a hermetically sealed door set up across the passage to the shaft station. Men's access to the pumps is through the pipeline way from the shaft.

3. *Underground locomotive shed.* Its arrangement depends on whether it is intended for trolley or storage-battery locomotives (Figs 89 and 90). In the latter case the shed has special stands for battery charging (*charging* room).

4. Both storage-battery and trolley locomotives require direct current and since mines are now supplied with three-phase alternating current it becomes necessary to provide *converter-station*

*rooms* for machines converting alternating current into direct (mercury arc rectifiers, synchronous converters). Fig. 90 illustrates general arrangement in converter room *1* near locomotive shed *2*, while a more detailed layout of equipment in one of such rooms is shown in Fig. 91.

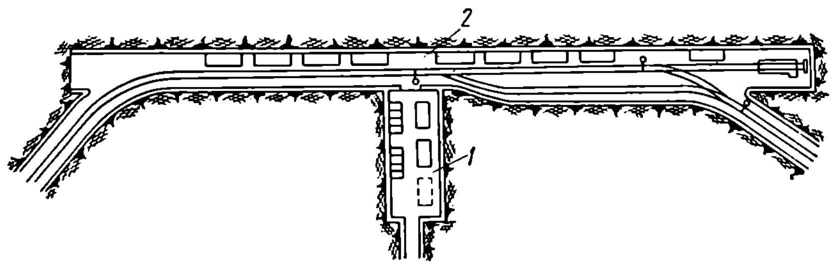



*Fig. 90.* Underground shed for storage-battery locomotives

It should be added that *condenser* electric locomotives operating direct from a. c. mains are being put into operation.

5. Three-phase current is supplied underground by high-tension cables (3,000-6,000 v). Inasmuch as most of the underground loads operate at a lower voltage (usually 250 or 380 v), this necessitates building *special rooms for underground electric substations* with step-down transformers and switching equipment. Very often these substations are set up in the immediate vicinity of pump rooms in

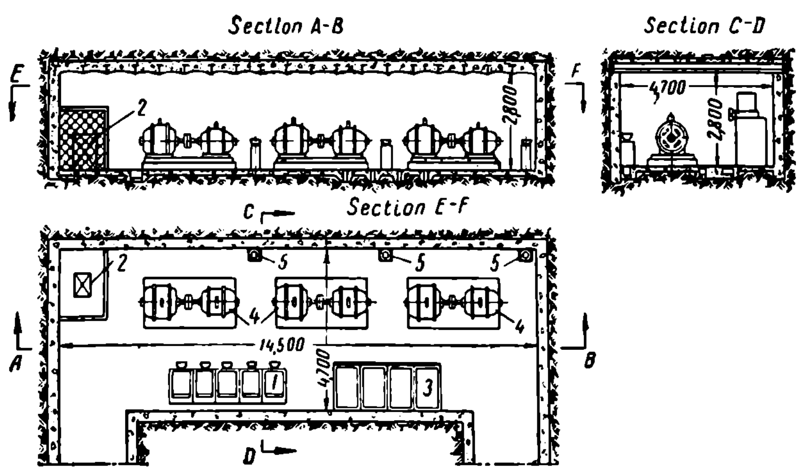



*Fig. 91.* Underground converter room

*1*—high-voltage distribution box; *2*—transformer; *3*—d.c. boards; *4*—motor-generator set; *5*—starting rheostat order to give them the same protection from flooding as in the case of pumping equipment.

6. *Storage room for fire-fighting materials and equipment* (Fig. 92) contains the necessary tools (shovels, picks, crowbars, sledge hammers, axes), fire-fighting equipment (fire extinguishers, water sprayers, pumps, hoses, barrels for water, pails), tarpauline, ventilation tubes and materials for rapid construction of bulkheads and seals (clay, bricks, boards, cement, sand). Some of these materials are kept in mine cars.

7. *Central control room* (Fig. 93) is the underground office of the dispatcher whose duty is to regulate underground haulage and the work at the stopes. In this he is assisted by a telephone operator.

8. At mines employing more than 700 men underground there is a *waiting* or *rest room* for miners waiting for cages to take them up or electric trains to bring them to the working faces. This room has wooden benches with backs and is connected with the workings of the shaft station by two passageways.

9. Underground *medical station* extends first aid to miners sustaining injuries while at work and provides ambulatory treatment of minor injuries or ailments for which miners are not granted sick leave. It has two rooms—one is the reception and also waiting room, the other a dressing room with appropriate equipment. In the cold season the temperature is kept up by electric heaters. A first-aid station is a must for all mines with no fewer than 1,000 underground workers on the payroll.

10. When a shaft is to be deepened to a new working level and there is no adequate space around the shaft station for setting up a stage hoist, a *special room* for it must be excavated.

11. Of particular importance are the underground *powder rooms.* They may be of *alveolar* (Fig. 94*a*) or *chamber* (Fig. 94*b*) type. In the former, explosives and blasting supplies are stored in small individual niches (cells) arranged in staggered rows on both sides of the room. In the latter case, explosive and blasting supplies are stored in separate chambers. The purpose of individual premises and arrangement of the rooms is explained by the caption to Fig. 94.

The unshaded space in the rooms in Fig. 94 should be provided with fireproof timbering. The arrangement of underground powder magazines and their location in relation to other underground workings must conform to the Safety Rules.

There are a great many ways of arranging the above-cited rooms within the area of the shaft station. In general, the decisive role in the planning of shaft stations is played by the conditions determining the operation of haulage and hoisting facilities, and the principal layout patterns should be elaborated to suit them. There are no essential difficulties in planning the location of other under-

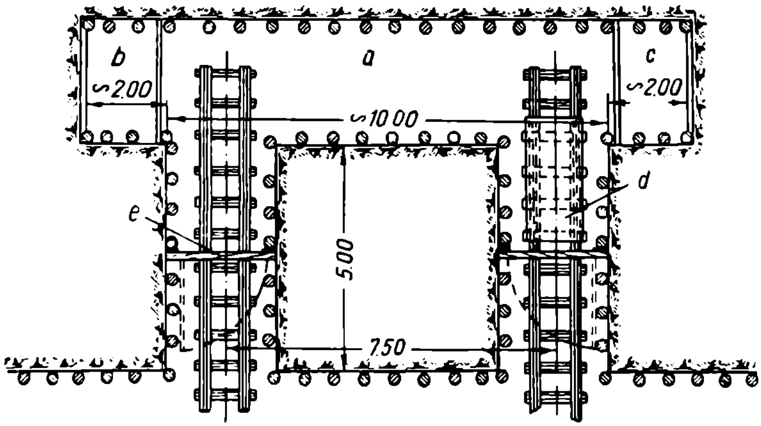



*Fig. 92.* Underground storeroom for fire-fighting materials and equipment

*a*—storage space for materials and instruments; *b*—storage space for clay and sand; *c*—storage space for bricks; *d*—mine-car; *e*—door

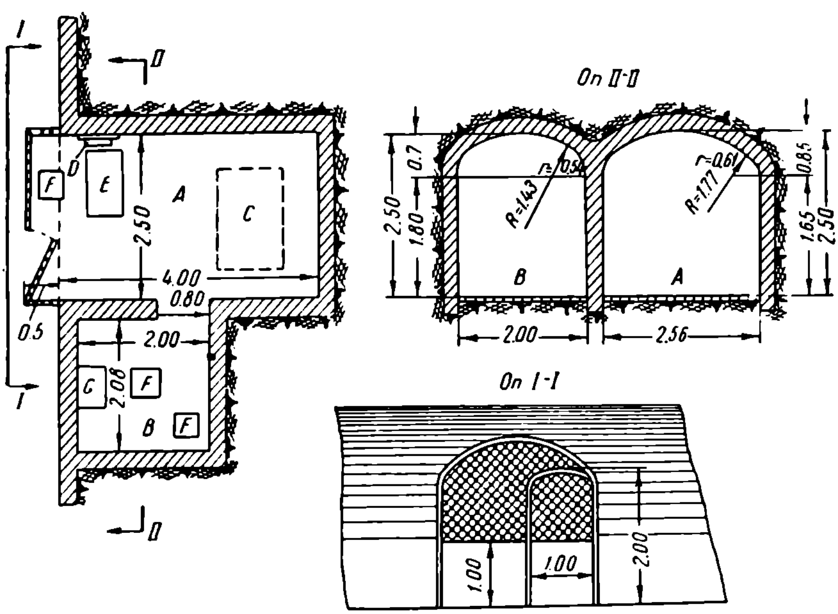



*Fig. 93.* Underground central control room

*A*—dispatcher's office; *B*—telephone operator's room; *C*—control desk; *D*—telephone; *E*—desk; *F*—chair; *G*—commutator

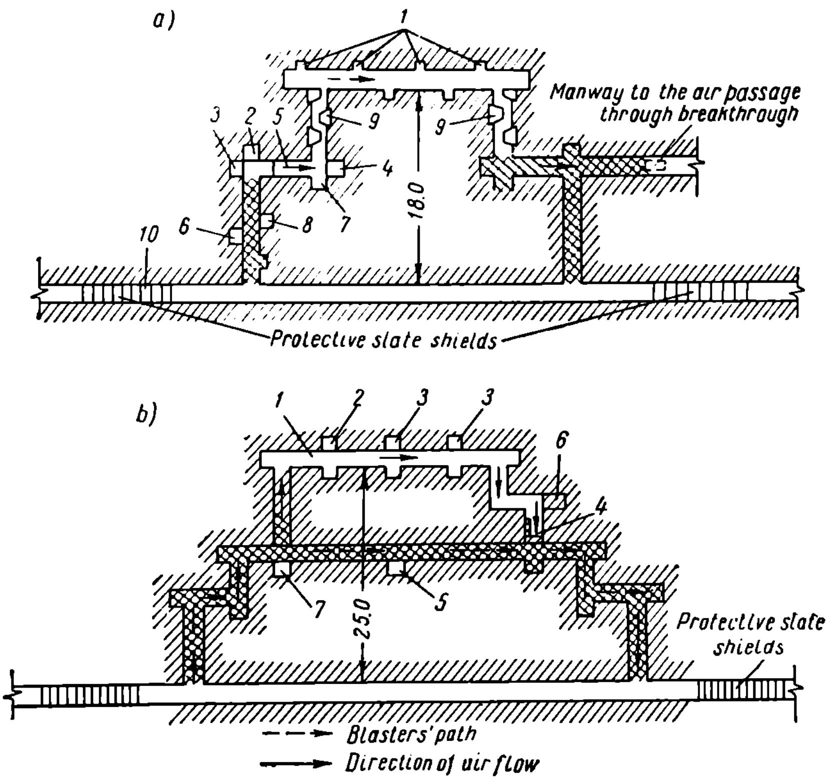



*Fig. 94.* Underground powder magazine

a—of alveolar type: 1—cells; 2—electric detonator checking room; 3—protective shell fitting out room; 4—igniting fuse storage room; 5—distribution room; 6—storage space for blasters' bags; 7—heating and signalling room; 8—watchman's bay; 9—crosspieces; 10—slate seals

b—of chamber type: 1—working adjacent to powder rooms; 2—blasting supplies storage room; 3—explosives storeroom; 4—blasting supplies distribution; 5—storage space for blasters' bags; 6—room where capsule detonators are connected with the igniting fuse and electric detonators checked; 7—space for electric heater

ground rooms, which can be arranged in a variety of ways. To facilitate the use of underground service rooms and reduce to the minimum the amount of excavation work, the electric substation should be together with the pump room; the underground locomotive shed with the battery charging station and the repair room; the central underground control room with the first-aid station. Figs 82 and 83 depict the layout of various service rooms in the shaft station area.

Since the underground service rooms with machines and equipment also contain lubricants and are extensively wired, they are classified as premises exposed to fire hazards and their protection, aeration and lighting are regulated strictly by Safety Rules and Rules for the Exploitation of Mines.

*Part Two*

# UNDERGROUND MINING OF MINERAL DEPOSITS

CHAPTER VII

# BASIC CONCEPTS AND TERMINOLOGY

## 1. Definition of Concept "Method of Mining Deposits"

As shown above, the work done in connection with direct extraction of the bulk of the mineral from a deposit is called *stoping operation*. Mine workings where stoping operations are conducted are called *rooms* or *stoping workings*, while the faces at which they are performed are called *stopes* or *working faces*. Excavating the valuable mineral at the stopes is usually referred to as *stoping*.

But before stoping operations can be begun in any part of the mine field, a more or less complex network of mine openings has to be created in order to open access to the stoping areas from the permanent mine workings made earlier in opening up a deposit. Subsequently, these openings will serve as communication ways for men, haulage, ventilation, etc. Mine openings of this type are called *development workings*, their faces—*development faces*, and the operations connected with the making of development openings or excavations—*development work*.

The latter has to follow a definite sequence in time and space and should, moreover, definitely precede the stoping operations and be properly coordinated with them.

This *definite sequence or order of driving development and stoping workings, coordinated in space and time, is termed method or system of mining a mineral deposit* (*or part thereof*).

A *proper* or *adequate* mining method is the one that maximally and simultaneously ensures three basic requirements: *safety of operations, economic efficiency and minimal losses of the useful mineral.*

In conditions prevailing in the Soviet socialist economy requirements calling for industrial safety and occupational hygiene are self-evident.

Economic efficiency of a mining method is evaluated by the minimal consumption of labour, mechanical energy and materials required for the extraction of the mineral from a deposit. As a whole, mining is a *labour-consuming* process and, therefore, in planning and applying any mining method particular emphasis should be laid upon the *highest possible efficiency of labour*. This major prerequisite is ensured mainly by *mechanisation of mining operations*, their proper organisation, introduction of advanced methods of work and adoption of a mining method that is most expedient and constructive in any given set of conditions. High labour capacity is a decisive prerequisite for the economic efficiency of any given mining method, since in the overall cost of mineral production the cost of labour by far exceeds the expenditure on other items. This, however, does not mean that efforts should not be made maximally to reduce expenses under other headings, such as materials (timber, explosives, etc.), power (electricity and compressed air).

It has been stressed above (Chapter I, Section 5) that every deposit should be worked in such a manner as to minimise mineral losses.

## 2. Choice of Mining Method

There are many different factors that have to be considered and carefully weighed in choosing a mining method.

Of fundamental importance are the shape, size and spatial position of a deposit—its depth of occurrence from the ground surface, angles of dip, mutual disposition in the deposit of seams, veins and ore bodies in general.

Also of high significance are features common to any given mineral—its composition, the nature of distribution of useful components, gangue inclusions, hardness, jointing, cleavage. The same applies to the enclosing country rocks.

In the case of many deposits one should take due account of their water-bearing capacity. Some deposits are liable to spontaneous combustion, while others contain explosive (chiefly methane) or asphyxiant (mainly carbon dioxide) gases or inflammable dust, or rocks which, when drilled, produce fine quartz dust engendering silicosis.

Of paramount importance for mining is *all-round mechanisation* of operations. Hence, methods of mechanisation should be regarded, as we shall see in greater detail below (Chapter IX, Section 15), as one of the major factors influencing the choice of a mining method in each individual case.

Another important factor is the economic value of a given mineral: its abundance or, conversely, scantiness in nature and the possibility of utilising it in the national economy.

The effect produced by the factors above, taken separately or jointly, is to be established in the case of the basic types of mineral occurrences—coal, ore and others, and it is this order of presentation that we have adopted in the book.

## 3. Principles Underlying Mining of Deposits

The extraction of a mineral and the driving of mine workings in the earth's crust in general create excavated areas. From the standpoint of the effect of such excavations on the stability of overlying rocks there are three basic principles for mining deposits: 1) the first provides for permanent abandonment of support mineral pillars; 2) the second calls for stowing mined-out areas; and 3) the third envisages caving of the capping.

1. *Mining with permanent abandonment of support pillars with useful mineral* consists in the following (Fig. 95): during excavation, intact portions of the mineral are left systematically within mined-out areas *aa*, and these are called *support pillars bb*. The designation was chosen because they serve as supports for the overlying rocks.

In the case of sufficiently large areas of excavations, support pillars may be subjected to the pressure of the weight of the entire rock mass extending to the ground surface, and this fact must be taken into account when their size is estimated. When the size of rooms *aa* and support pillars *bb* is selected properly, there is practically no perceptible movement of overlying rocks and the ground surface above the deposit under exploitation actually remains completely unaffected.

Accurate measurements and theoretical calculations demonstrate that the support pillars which fully meet the requirements set before them and which reveal no signs of destruction in the form of fissures or visible changes in size are nevertheless subject to insignificant deformations of no essential importance for mining operations as such. One typical example of mining based on this principle is the production of rock salt in the Artyomovsk district (Donets basin) and potassium salt in Solikamsk (Northern Urals) (See Chapter XVIII).

2. With mining that involves *stowing* the mined-out areas are filled with packing materials—barren or waste rocks, sometimes smelter slags, concentration plant tailings, etc. (Fig. 96). Since the mine-fill is apt to *shrink* somewhat (that is, since its initial volume is apt to decrease), there is a possibility of certain shifts of rocks overlying the mined-out and filled areas. These displacements assume

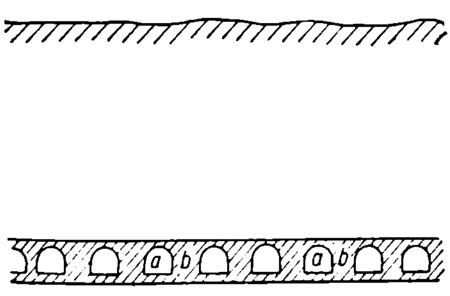


*Fig. 95.* Diagram showing mining with natural support pillars

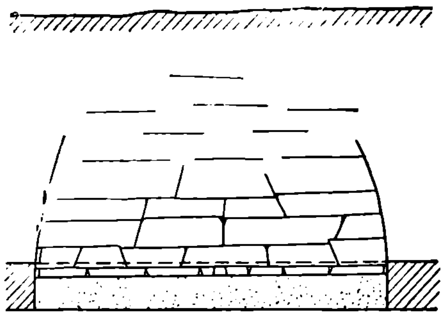


*Fig. 96.* Schematic representation of cut-and-fill mining

the form of small fissures or crevices and subsidence of rock masses that end at a certain level. If, because of the relatively small depth of the occurrence, the displacement zone reaches the ground surface, the latter may, to a certain extent, sink or sag. On the other hand, when the ground movement comes almost to a standstill at a certain depth, the surface lying over the mined-out and filled areas remains practically intact.

The problems of stowing mined-out spaces present considerable complications and are, therefore, discussed in Chapter VIII.

3. In mining with *caving*, that is, when the ground capping the mined-out area is not held in place either by pillars or by mine-fill,

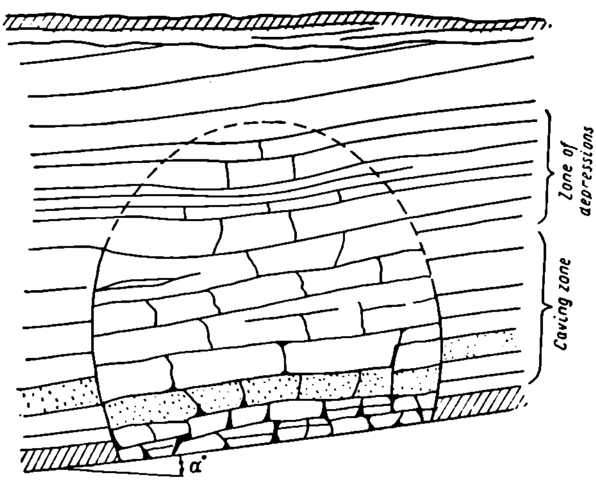



*Fig. 97.* Mining by caving

its movement, as a rule, is sharply accentuated (Fig. 97). A *zone of caving* appears immediately over the mined-out area, where the ground undergoes ruptures, is broken up by fractures, and where one can observe the caving of pieces, lumps and large blocks of ground. The appearance of ruptures and fractures and the displacement of individual blocks of ground tend somewhat to increase the summary volume of its mass. The increment may become equal to the volume of the mined-out space, after which the cavings will cease by themselves. But since the weight of the overlying strata will compress the caved ground, and these overlying rocks will tend to subside, there will be a *zone of depressions* (formerly known as a zone of settlement) forming above that of caving, and in this zone the ground will sag and develop fractures. The depression zone ends at a certain elevation above the mined-out space and the ground occurring above will not experience any shifts. If, however, the mined-out area is sufficiently extensive, the ground lying over the above-cited depression zone, intersected with crevices, may sag somewhat too, but gradually and without developing any fractures. The volume of the ground involved in this process is usually designated as a *zone of smooth sagging*.

In mining involving caving systems, there are three typical conditions the ground surface may find itself in:

1. The caving zone extends to the surface. In this case, the movement of ground, particularly when the thickness of the deposit is considerable, may be extremely marked—the ground surface is broken up by fractures and there appear holes and pits, sometimes of enormous size.

2. The ground surface is reached by the depression zone only. This is manifested by its subsidence and sagging.

3. The ground surface lies above the zones of caving and depression and is not subject to displacement (deformations) of any practical importance (though some very slight movements may sometimes be registered by instruments).

The three basic principles of mining, viz: with support pillars, mine-filling (cut-and-fill system) and caving, are employed now in their pure form, now in combination with each other.

Very often pillars of useful mineral, supporting wall rocks, are left *for a time* and not permanently, and after their recovery the wall rocks begin to cave in.

Mined-out space may be filled with gob partially, in some sections, the rock between which may cave in.

Sometimes the mined-out areas are *provisionally filled* with broken mineral, which is later taken out of the mine. This temporary abandonment of the mineral is called *shrinkage stoping*.

In mining involving caving, the stopes are protected by different types of *timbering*. In most cases, the timbering of stopes or working faces is also necessary in mining involving filling worked-out areas. It is only in mining with abandoned support pillars that no timbering is employed in the stopes.

The purpose of the timbering, filling, support and other pillars is to protect mine workings from the impact of *rock pressure*.

The main factor behind rock pressure is the weight of the rock mass, but in individual instances it may also be the compressive force developed by water and gases in the rocks and the force engendered by changes in the components of rocks (for example, the force developed by swelling argillaceous rocks following absorption of water, etc.).

*Rock pressure* manifests itself by the subsidence, sagging, caving and other movements of ground, development of fractures, cracks, etc. These phenomena may be attended by deformations of mine workings, and it is to avert them that it becomes necessary to install timbering, resort to filling or gobbing, or leave support pillars with useful mineral.

Thus, *rock pressure is the compressive force developed in and by the wall rocks surrounding mine workings, and its presence requires installation of timbering (or other supports) to prevent deformations in these workings.*

Implementation of various measures, such as timbering, filling, abandonment of pillars, controlled roof caving (see Chapter X), etc., directed at eliminating the harmful effect produced by rock pressure or at changing its character, is known as *pressure control.*

From this standpoint, rock pressure is regarded as a natural force that must be managed, or *controlled*, with the aid of technical devices. However, there may arise a question of *utilising* the forces of rock pressure for industrial purposes. In fact, below we shall come across instances when this pressure, for example, is helpful in *squeezing* coal, thus facilitating its extraction, and learn of mining systems providing for *undercutting* the ore body so as to utilise the force of gravity in breaking, transferring and drawing the ore.

## 4. Drawings of Mining Systems

Like in Fig. 3, drawings of mining systems may be: 1) *plans*, that is, projections on a horizontal plane; 2) projections on a *vertical plane* running along the strike of a deposit; 3) projections on an inclined plane, depicting the spatial position of a deposit; 4) vertical section along planes, more often than not in the direction running across the strike.

The basic type of drawings of the systems of mining is projection on an inclined plane, since in this case dimensions along the lines of strike and dip are not distorted. If the position of the deposit is inconsistent or irregular, these drawings appear to be somewhat schematic.

Underground survey drawings or mine maps are plans, or, in steeply dipping deposits, projections on a vertical plane. Vertical sections are appended to mine maps.

CHAPTER VIII

# FILLING

## 1. The Significance of Filling

The mined-out area may be *filled* after the extraction of the useful mineral. The materials for mine-fill are waste or, in rare instances, smelter slags or concentration mill tailings.

The operation of placing or arranging mine-fill is usually designated as *filling*, stowing or gobbing up of the goaf.

Mining of mineral deposits with subsequent filling or gobbing has a number of major advantages. One is the drastic reduction of the intensity of shifting taking place in the rock strata overlying worked-out and filled areas (in high-dipping deposits this also applies to wall rocks). The chances of caving in of the rocks in the stopes which endanger men's lives and impair operations are reduced substantially. Filling tends to decrease the pressure produced by country rocks on the mine workings and cuts down consumption of timber and mineral losses in pillars. Since leaving self-igniting minerals (most types of coal, pyrites) underground entails fire hazards the cut-and-fill method of mining also has the advantage in that it eliminates or, at least, tends to reduce the danger of mine fires. With mining methods involving filling of goafs, it is appreciably easier to arrange proper ventilation, since there is no air leakage through crevices in rocks, which is the case with the caving system and which it is difficult to do away with. They also ensure better maintenance of surface mine structures.

This brief enumeration of the basic advantages of the cut-and-fill methods of mining is illustrated in greater detail below.

The application of the methods under discussion, however, requires setting up of special and complex *installations for filling operations*. While in a mine employing no fill all the operations are in the final analysis directed at ensuring the production of mineral in stopes or at working faces, its haulage underground and surface transportation to the ultimate points of shipping or processing, the cut-and-fill method provides for another series of analogous processes demanding special arrangements.

For example, in working coal deposits with mine-fill, the quantity of produced coal and the amount of the stow required are approximately the same in weight. When large volumes of the stow are required, it is obtained on the ground surface.

This implies: 1) opening up a quarry for the winning of mine-fill, equipped with installations for its crushing, sizing and mixing; 2) transportation of these materials from the quarry to the site where they are lowered into the mine; 3) the lowering of the mine-fill into the mine; 4) its haulage underground to the faces or stopes; 5) filling in or stowing of mined-out spaces.

This chain of operations requires installing special equipment and manpower. To reduce the amount of labour required for these operations, they should be thoroughly mechanised.

Hence, the *economic aspect* of the issue is of extreme importance when deciding whether any given deposit is to be worked with or without mine-fill. This complex question should be discussed in all its details. To form an appropriate judgement on the economic results of the method in question, it is necessary to compare, in each individual instance, both the technical and the economic advantages and shortcomings of working the deposit with and without mine-fill, and take into account all the operations involved in filling, the methods of mining used, mineral losses, etc.

As will be seen below, the filling is of the greatest importance in mining thick, steeply dipping deposits, particularly of self-igniting coal and pyrites.

## 2. Types of Filling

If the mine-fill is placed throughout the entire worked-out area, it is called *whole* or *complete* fill; if not—*partial* or *incomplete*.

It is common to classify filling according to the methods employed for its arrangement. Correspondingly, filling is subdivided into *hand-stowing* and *filling by flushing*, *mechanical* filling, *pneumatic* and *hydraulic* or *float* fills.

Hand-stowing is now employed only in the arrangement of rib-fills in mining thin, flat-dipping seams for roof control or protection of workings from rock pressure. These operations are described in Section 4 of Chapter XI.

With the hydraulic method the mine-fill is delivered to the gob area via pipelines by a jet of water. All types of filling other than hydraulic are termed *dry*.

## 3. Sources and Properties of Mine-Fills

Stows can be obtained either in underground workings or on the ground surface.

1. *Underground sources of mine-fills.* In some instances the stow may be procured at the working face or in the stope itself. In working very thin ore deposits to obtain sufficient head room in the stope, it is necessary to blast country rocks whose volume is more than enough to complete filling and part of them have even to be taken out of the stope area because of the increased volume of the broken ground, characterised by the coefficient of expansion. In working ore veins, barren rocks that can be used for filling may be separated in the stope. In mining coal seams, bands or interlayers of gangue, false roof or bed bottom can be used as mine-fill, if they do not contain self-igniting carbonaceous matter.

The common source of fill are the wall rocks blasted in driving mine workings. In mining thin coal seams, after the faces have been worked, it is a frequent practice to push forward special *gob* or *lateral entries* for the sole purpose of procuring the necessary mine-fill. Much stow can also be obtained when driving the main mine openings (shafts, rooms, crosscuts) and when developing new levels. A certain amount of rock is gained when repairing mine workings.

2. In mining with large fills the above-cited underground sources usually prove to be insufficient and the stow has to be excavated *on the ground surface.* For this purpose special *borrow pits* are arranged and the rocks used for filling are won by methods applied in open-cut mining (see Chapters XXV and XXVI).

Rocks procured in a borrow pit are best used for filling without undergoing any additional processing.

For example, sand with a slight admixture of clay particles is considered to be the best material for hydraulic fill. If the borrow pit with such sand is located in the vicinity of the site where the stow is lowered into underground workings, the sand can be won and transported to this site by hydromechanical methods (see Chapter XXVI).

Lumpy rock (not exceeding 60 mm) can also be used for hydraulic fill. Lumps of 60-70 mm can be used for pneumatic filling. Larger lumps may be used for filling by flushing, but their size is limited by transport facilities and the possibility of bringing them down into mined-out areas without endangering the safety of timbering. If it is bedrocks (hard rocks) that are mined in the pit for mine-fill, requiring blasting and loading of large-sized lumps by power shovels, then it is necessary to set up *crushing* plants to obtain the filling material.

Crushed rock or sand is delivered to the site where it is lowered into underground workings by rail, in cars provided with automatic discharge facilities. In areas with mountainous topography cableways can be used instead of railway tracks. When transport distances are short, conveyer plants may be employed.

Apart from the rocks excavated in pits, mines can be filled with such loose or lumpy mineral materials as rocks from old waste dumps, mill tailings from concentration and chemical plants, smelter slags, etc., if conditions permit it. All these materials should not be self-igniting. In mining potassium deposits, the waste left after the processing of salts won in the mine is employed for filling. In many copper mines of the Urals granulated slags obtained in copper smelting by pouring molten slag into water are used as hydraulic fill (generally, the term "granulated slag" is applied to a fine-grained material obtained from smelter slag when it is poured in molten state into water or broken by air or steam blown through its mass).

Dry mine-fill may be lowered into underground workings by gravity or in mine cars. Lowering by gravity is effected through dumping chutes or special pipes. *Dumping chutes* are arranged in inclined openings (with the slope angle not less than about 45°). Receiving mill holes are arranged over their mouths for the waste discharged from transport facilities (mine cars, conveyers), while below there are drawing chutes for the discharge of waste into underground transport vehicles. To prevent men from falling and preclude penetration of excessively large blocks, a solidly built grate with openings corresponding to the size of lumps is installed over the mouth of the chute. The grate is provided with a cover to protect it from rain and snow. If the descending material is sharp-angled, as is the case with crushed rock, the walls of the chute should be lined with thick boards or reinforced with rails. The fill can also be lowered into underground workings through *vertical pipelines* made of thick-walled (8-10 mm) wear-resistant steel pipes with a diameter of 250-300 mm and consisting of sections 30-50 metres long with clearances between them to prevent formation of "air plugs" and to facilitate repairs and replacement of pipes. To obtain the necessary clearances or plays, the upper end of each section is provided with a bell-shaped funnel into which the lower portion of the overhead section is inserted so that it does not come into contact with the former. In the mine at the bottom end of the pipeline a reception chamber is arranged with a "cushion" made of rocks to absorb the shock caused by the velocity of the descending mine-fill.

The stow can also be brought down in cages, loaded in ordinary mine cars or skips.

Underground dry mine-fill is transported to stopes, working faces or pneumatic installations in mine cars or by conveyer.

3. When it is packed into mined-out space, the fill occupies a larger volume than it did at the site of its original occurrence. The volume of a solid rock increases when it is broken into separate pieces. The ratio between the new volume and the initial is called *coefficient of volumetric expansion.*

| Types of rock | Coefficient of volumetric expansion |
|---|---|
| Running fine sand | 1.05 |
| Sand, gravel | 1.1-1.2 |
| Sandy loam, loam, soft clay | 1.2-1.25 |
| Marl, sod | 1.25-1.3 |
| Hard, heavy clay, hard marl | 1.25-1.35 |
| Rocks: | |
| soft | 1.3-1.4 |
| hard | 1.4-1.5 |

Natural or artificial friable, loose or lumpy materials (sand, gravel, waste from dumps, granulated slags, etc.) are already loose at the site of their occurrence, and when extracted for filling their coefficient of expansion is low.

Even with the so-called complete or whole fill, the volume that has actually to be packed is less than that occupied by the mineral "in situ", since usually there is some timber left in the mined-out areas, which takes up a portion of the volume and prevents compactness of fill. Furthermore, certain mine workings are sometimes left unpacked within the block to be filled. In some instances, by the time filling is completed, the volume of the mined-out space manages to shrink due to the settling or heaving of the surrounding rocks. The gentle dip makes it rather difficult to pack the space immediately adjacent to the roof sufficiently tightly. Hence, to fill a certain volume of the mined-out area, even if it is to be completely packed, the amount of the stow needed is 1.5-2.5 times less than its original volume in place. The lower figure refers to loose ground, thicker deposits and steep dip.

The above also explains why dry mine-fill is capable of being compressed by rock pressure on being placed into a mined-out space. It is understandable too why this reduction in volume (*shrinkage*) is greater in flat-dipping deposits than in the steeply inclined, and greater with large-sized lumps than with fine-grained material (see Table 2).

Consequently, unlike hydraulic or pneumatic filling, the dry pack may result in quite considerable shrinkage.

*Table 2*

**Reduction in the Volume of Mined-Out Space with Different Methods of Filling**

| Types of fill | | Ratio between mined-out space and the thickness of a bed after shrinkage of the mine-fill, in per cent |
|---|---|---|
| Hydraulic fill | | 85-95 |
| Pneumatic fill | | 80-90 |
| Filling by flushing in high-dipping beds | fine-grained rock | 75-85 |
| | lumpy rock | 60-75 |
| Dry fill in flat-dipping deposits | with the mine-fill supplied to the stopes from without | 40-60 |
| | with the source of fill situated in the mine field | 15-30 |

## 4. Filling by Flushing

Filling by flushing is distinguished by its movement *by gravity* in the mined-out area. Hence, this type of fill is employed only in high-dipping deposits. The fill is supplied from an entry or drift lying overhead of the given stope. In the case of filling by flushing, the worked-out space may be packed wholly or partially. In the latter instance, *rib-fills* are arranged either on the strike or across it (along the dip). The rib-fills or side packs extended on the strike often serve to protect the entries.

In high-dipping deposits, the fill rests on *gate-stulls* supported by heavy props to avert its slipping down. The dip rib-fills can be held in place with the aid of stop boards. With complete filling by flushing, the fill in the stoped-out areas is placed either at the angle of its repose, which corresponds to the pitch of the bed, or else is held in place by special stop boards. In steeply pitching beds, the compactness of the fill, especially when it is made up of small-size lumps, is quite satisfactory (see Table 2).

One variant of the filling by flushing is the mobile fill, *transferred* from an overlying mine workings packed in the past. This method has serious shortcomings.

## 5. Mechanisation of Filling Operations

Packing of fill by hand presents considerable difficulties. The filling operation can now be mechanised by several methods:

1) by machines in which the required velocity can be imparted to the mine-fill by a fast-moving belt, rotating blade-wheel, etc. Such machines are known as *fillers*, while stowing done with their aid is termed *mechanical*;

2) by supplying the fill with an air current (*pneumatic* filling);

3) by delivering the fill with a water jet (*hydraulic* or *float* fill);

4) by using *scrapers*, which have been tried on many occasions for mechanising filling operations, particularly with barren rock from gob or side entries, but this method has failed to win recognition.

## 6. Mechanical Filling

Two operative principles have been suggested for filling or stowing machines. Those operating on the basis of the first *throw* the mine-fill (*throwing machines*), while the ones built on the basis of the second stow the fill with the aid of special inclined conveyers which bring it to the roof of the stope and simultaneously compact or ram it (*compacting stowing machines*). The latter type of filling machines has found no practical use.

The operating mechanism of the stowing machines that throw the fill is a blade wheel or an endless belt.

A diagram illustrating the action of filling machines of *belt-conveyer* type is shown in Fig. 98. The fill is usually fed by a conveyer to a hopper, whence it slides down onto a high-speed endless belt which imparts to it the velocity required for throwing. This velocity is communicated to the material through its friction with the belt. To increase this friction the belt is made curved and to assume this shape it passes round a drum provided with flanges. The centrifugal force appearing thereupon tends to enhance the adhesion of the fill to the belt and hence the friction force.

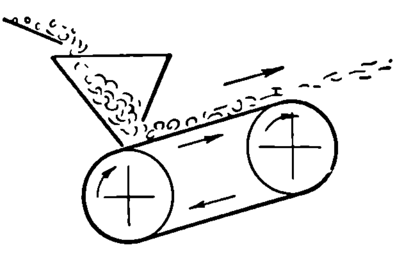


*Fig. 98.* Operating principle of a belt-type mine-fill throwing machine

This machine, designed by the Kuznetsk branch of the State Institute for Designing Coal Equipment, is shown in Fig. 99. Its operating characteristics are: capacity—60 $m^3$/hr, width of the belt—500 mm, speed of the belt—15 m/sec, angle of slope—from 18 to 30°, throwing distance (at the slope angle of

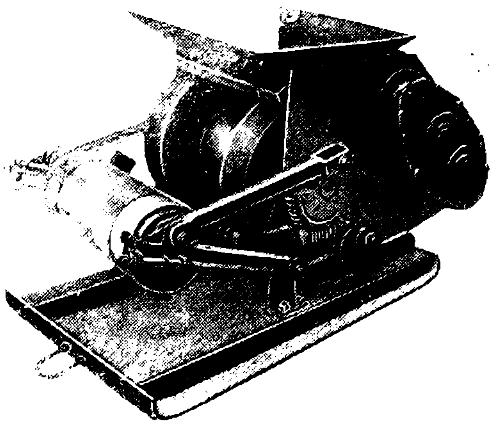


*Fig. 99.* Throwing machine

30°)—6.3 metres; overall dimensions of the machine: length—1.7 metres, width—0.8 metre, height—925 mm; weight of the machine (without motor)—925 kg; rated power of the motor—11.5 kw at 1,500 rpm.

Thus, the distance to which these machines can throw the fill is rather restricted.

A very important feature of these machines is their ability to bring the pack up to the roof of the stoped-out area. The mine-fill is fed to the machines by conveyers. In order to reduce the number and duration of operations in the stope, which involve transfers of the throwing machine, it is recommended to feed it with the fill from belt conveyers provided with special devices for their rapid shortening.

## 7. Pneumatic Fill

Stowing units operating with compressed air may be subdivided into two categories:

those in which the fill is transported by compressed air through pipes running along development openings and working stopes and is then thrown into the mined-out space;

those where compressed air is employed only for throwing the fill into worked-out space (stowing machines of the *ejector* type). In purpose, the units of the latter group are similar to the throwing machines described in Section 6 of this chapter, but instead of a bladed wheel or belt the fill is thrown out by a compressed air current. These machines, however, have failed to find practical use.

With the aid of compressed air the mine-fill can be delivered via pipes to various distances, which in the most favourable conditions may be as much as 800-1,000 metres.

The phenomena characterising the movement of the fill in pipes entrained by a current of compressed air are very complex indeed. With a sufficient velocity of the air the finer particles can be caught up by the air current and transported in suspended state (*floating*). Larger particles can roll near the inside walls of the pipes and, being subjected to a reciprocal impact, move forward in leaps. Inasmuch as the number of particles carried by the air current is extremely high and their size varies within a very wide range, the phenomena of floating, rolling and leaping of particles actually form combinations of a very complex type.

In modern pneumatic stowing units the permissible size of particles handled should not exceed 70-80 mm.

Tests have proved that the diameter of stowing pipelines should be about 150-200 mm if the formation of "plugs", that is, clogging, is to be prevented.

The mixture of compressed air and fill particles moves along the pipes at extremely high velocity. At the charging end of the pipeline this velocity may be around 20-40 m/sec, depending upon the properties of the fill, but after that it increases somewhat and, in pipelines of up to 400-500 metres, the velocity at the discharge end becomes as high as 50-80 m/sec, while longer pipelines raise this "terminal" velocity still higher.

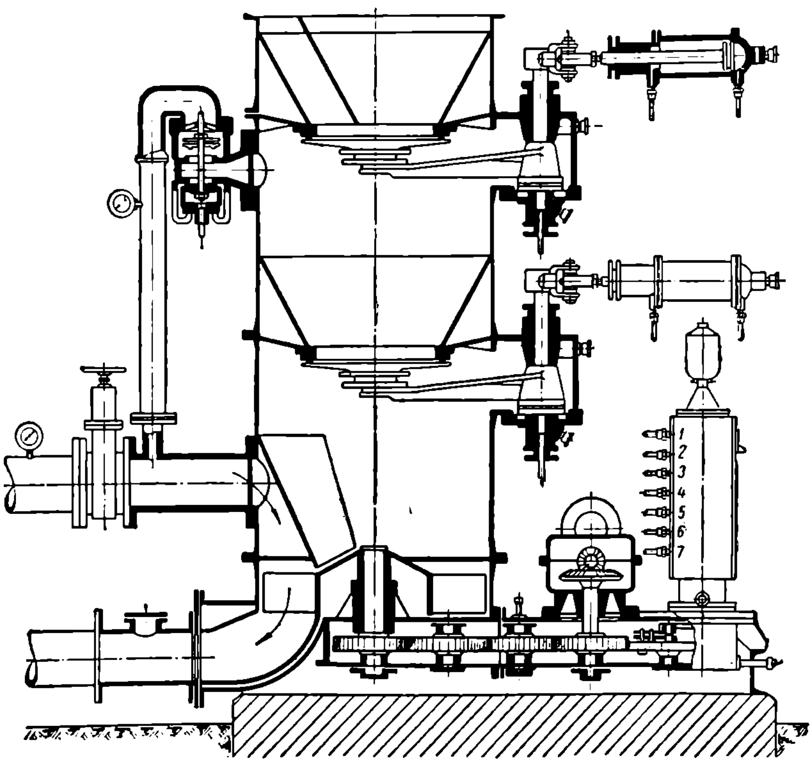



*Fig. 100.* Diagram showing the arrangement of a double-chamber air-blast stowing unit with automatic control

The transport of fill particles along the pipes by compressed air currents is accompanied by head losses and the air should be fed to them under a pressure of from 2.5 to 4 atmospheres, depending upon the type of the stowing units.

From the standpoint of operational safety and increased efficiency of the air-blast stowing machines, the best stow should include crushed and sized soft rocks, such as clay and sandy shale. Harder rocks—for example, crushed sandstone—cause rapid wear and tear of pipes.

Considerable amounts of fines (with particle size below 10 mm) tend to reduce the efficiency of the unit. The humidity of the fill rock should not exceed 3 per cent for otherwise minute particles may adhere to the inside walls of the pipes. Clay is not suitable for pneumatic fill.

Among the large number of stowing machine types proposed, the most widely used are those of chamber and drum types.

Fig. 100 gives the diagram and Fig. 101 the general view of a *double-chamber air-blast goaf-stowing machine.* Its cylindrical body has two charging hoppers closed from below by two airtight disk-shaped gates. The mine-fill is fed evenly to the upper hopper, that is, by a conveyer or a special feeder, and not by a chute. When the gate is pulled out, the fill goes to the upper chamber, the bottom of which also has the shape of a funnel provided with a gate. The gates are opened and closed alternatively by an automatic air-operated device.

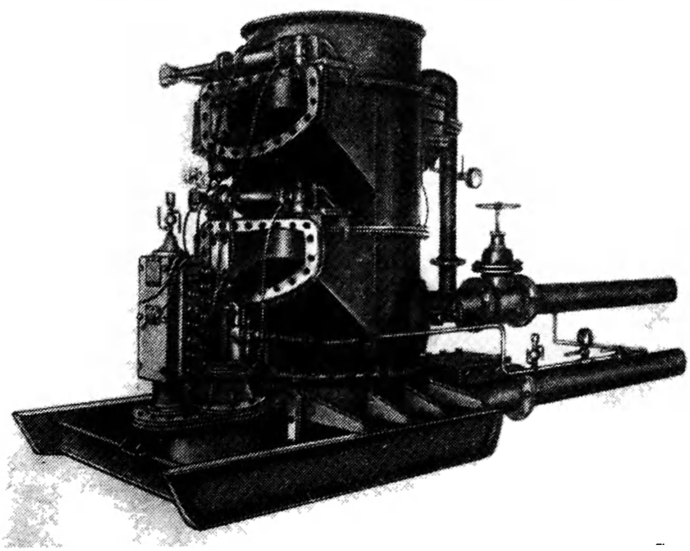


*Fig. 101.* Stowing machine with automatic control

From the upper chamber the fill pours down into the lower one at the time when air pressure in both chambers is the same. At the bottom of the lower chamber is a deflecting disk provided with vertical radial partition plates enclosing individual cells. The deflecting disk is mounted on a vertical shaft revolved by a motor with the aid of a set of transmission gears. When one of the cells is brought by the revolving disk to the pipe supplying compressed air, the fill is blown out from the cell and thrown by the air blast into the pipeline delivering the fill to its place of destination. The unit is provided either with electric or air drive.

A stowing machine of this type, designed by the Kuznetsk branch of the State Institute for Designing Coal Equipment (model П3М-1) and manufactured by the Kiselyovsk (Kuznetsk Basin) Engineering Works has the following operating characteristics: air pressure—3-4 atm; output—35 $m^3$ hr; maximal size of individual fill lumps—up to 80 mm; distance of the mine-fill delivery—up to 400-600 and even 800 metres; air-motor capacity—around 10 hp; overall weight of the unit—about 3 tons; bores of pipes: air-pipe—100 mm, stowing pipe—150 mm; air consumption: for the operation of the motor—4.5-5.5 $m^3$/min, per 1 cu m of the mine-fill—100-160 cu m; overall dimensions of the machine: height—2.15 metres, width—1 metre, length—2.5 metres.

A diagram of another air-blast stowing machine—with a measuring *drum* or *cylinder*—is depicted by Fig. 102. A drum with radially placed partition walls turns inside a horizontal cylinder. The mine-fill is poured into the unit through a charging hopper from above and leaves it via a spout, whence it is caught up and driven into the pipeline by a jet of compressed air. To secure the operation of this machine a minimum clearance (of not more than 0.5 mm) is needed between the cylinder and the partition walls of the drum.

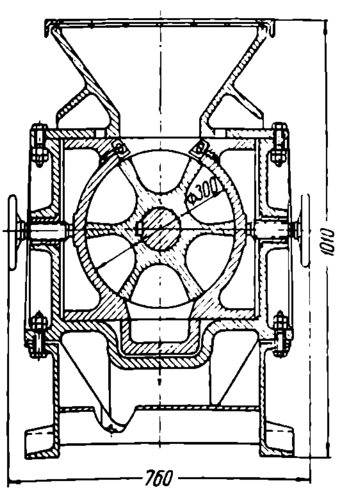


*Fig. 102.* Stowing machine with a measuring drum

A machine of this type, designed by the Kuznetsk branch office of the Institute for Designing Coal Equipment (М3ПМ-1—small air-blast stowing machine), is characterised by the following operating data: output—38 $m^3$ hr; distance of the mine-fill delivery—up to 400 metres; air consumption per 1 cu m of the fill—80-160 cu m; air-motor capacity—10 hp; air pressure required: in the pipeline of the motor — 4.5-5 atm; in the stowing pipeline—2-2.5 atm; overall dimensions of the unit—length —2.65 metres, width—0.67 metre, height—1.01 metres; weight of the unit—1.73 tons.

The air-blast filling machines described above are set up underground, in the vicinity of working sections. The rock for the fill is supplied by conveyers or in mine cars. In the latter case (Fig. 103), mechanical feeder *2* is set up beneath bin *1*, from which the fill is delivered evenly to short belt conveyer *3*, and then to stowing machine *4*.

Machines of the drum type are simpler in design and service and smaller than chamber units. But they are suitable for shorter distances and, to prevent excessive wear, require softer rocks for the fill.

From the air-blast units the mine-fill is delivered to the working areas by special pipes and is thrown out into the mined-out space through a nozzle. The machine is operated by a man provided with protective goggles. To make shortening of pipes in the stope convenient, use may be made of a telescopic arrangement (Fig. 104), which includes extensible sliding pipes *1* and *5*, whose position in relation to one another may be adjusted by flange *2*, holder *3* and gasket *4*. The stream of the fill may be directed where desired by deflector *1* (Fig. 105), set up at the discharge end of the pipe and fixed in position by means of rod *2*, handle *3* and toothed sector *4*. The deflector can also be made to turn around the axis of the pipe.

Individual sections of the pipeline, removed along with the extension of the packed zone, are put down in the stope in an order facilitating the next filling cycle. Two-way signallisation is provided for between the stope and the stowing unit. In the event dust is produced during the filling operation, spray water is supplied into the pipeline, just before its discharge end. With air-blast stowing units it is possible to bring the mine-fill up to the roof of the working.

The line is made of seamless steel pipes. Those laid permanently in strike entries are 5-6 metres long, possess thick walls (8-10 mm) to slow down their wear, and are sometimes lined. To facilitate their handling, pipes used in stopes should be light and, therefore, about 2-3 metres long and have thin walls (3-4 mm). Flange connection requires about 6-8 minutes for each pipe. Therefore, special devices are used to enable to connect them in 1-2 minutes. One of these devices is shown in Fig. 106. Flange *1* is welded to one of the pipes to be connected and is provided with an annular lug which can be inserted into a slot in flange *2* of the next pipe. The connection is made airtight by rubber packing ring *3* in the slot. The pipes are braced by the cams of lever *4*, secured in the ears of clamp *5*.

Permanent (stationary) pipelines of average service-life pass through up to 60,000 cu m of clay shale, or 25,000 cu m of sandstone.

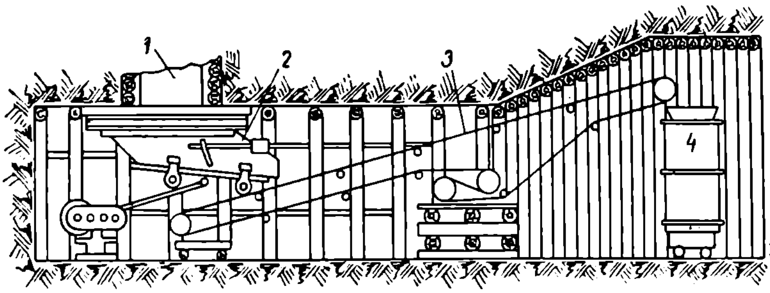



*Fig. 103.* Installation for uniform delivery of the fill to an air-blast stowing unit

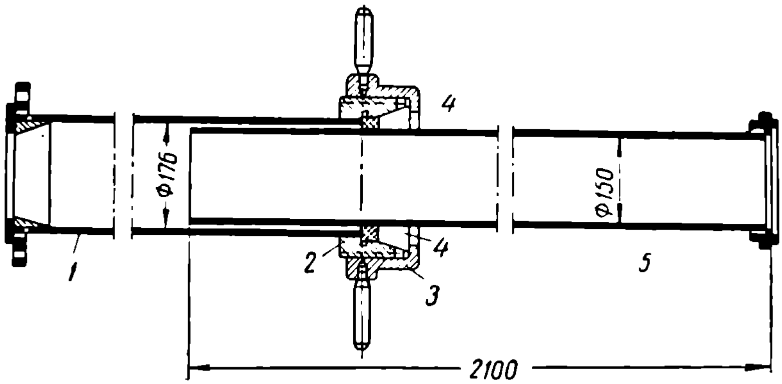



*Fig. 104.* Telescopic installation for shortening pipeline sections in the stoping area

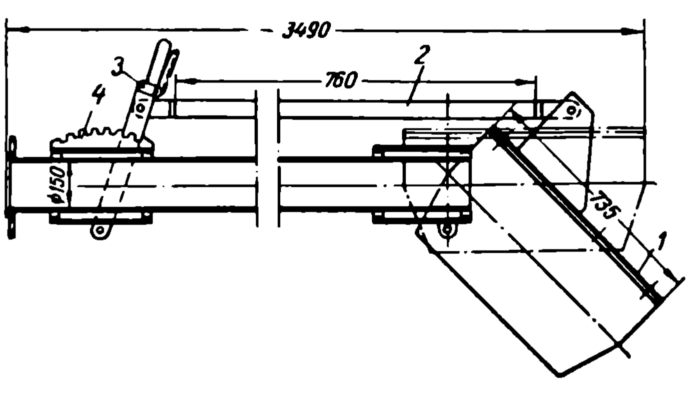



*Fig. 105.* Stowing pipeline deflector

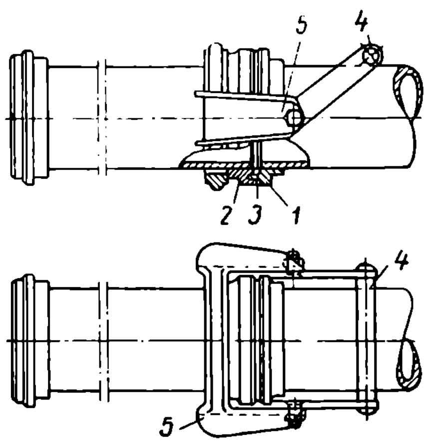



*Fig. 106.* Installation for rapid connection of stowage pipes

The portable thin-walled pipes employed in the stopes are liable to wear out much faster.

A pipeline should be suspended and not laid down rigidly, for suspended pipes, vibrating under the impact of mine-fill particles, are somewhat less exposed to wear.

To reduce the wear, pipes may be reinforced by lining of chrome pig-iron, cast basalt or other materials. Such pipes, however, are expensive, heavy (since their outer diameter must be larger) and inconvenient to handle. Besides, lining complicates detection of cloggings by drumming on the pipe.

Special measures should be taken to prevent increased wear at pipeline bendings. Fig. 107 shows a bending with a small angle of turn and made of two short pipe pieces, their diameter increased to facilitate reinforcing detachable steel plates.

Fig. 108 shows a bending made of two central *1* and two side pieces *2*. Its outer side, subject to maximal wear, is protected by inserts *5* of chrome steel. The central pipes have holes, provided with covers *4* with bolts and stiffeners *3* which hold the inserts in place. The arrangement of this type makes it possible quickly to withdraw or put in the inserts during repair work, or to eliminate cloggings in the pipes.

Experience has proved it superfluous to resort to large radii of bending in stowing pipelines. In pipes with a diameter of 150-200 mm, bendings with a radius of 450-600 mm and somewhat more may be used quite safely.

Air-blast stowing units are usually supplied with compressed air from the general air network of the mine. But since air pressure in the general network is greater than that required for air-blast stowing units (2.5-3.5 and up to 4 atm), the line with low pressure is connected to the main air network of the mine by special reduction valves. If the rated capacity of a compressor plant is as high as 200-300 $m^3$/min (of free air under atmospheric pressure), the units comprising it may be of a piston type. However, with higher requirements in air, turbocompressors are preferable.

The amount of electric power consumed by an air-blast stowing operation is quite considerable—usually 13-20 kwhr per 1 cu m of the mine-fill.

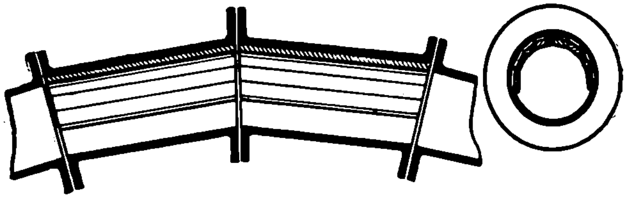


*Fig. 107.* Pipe lining at a bending in pneumatic filling

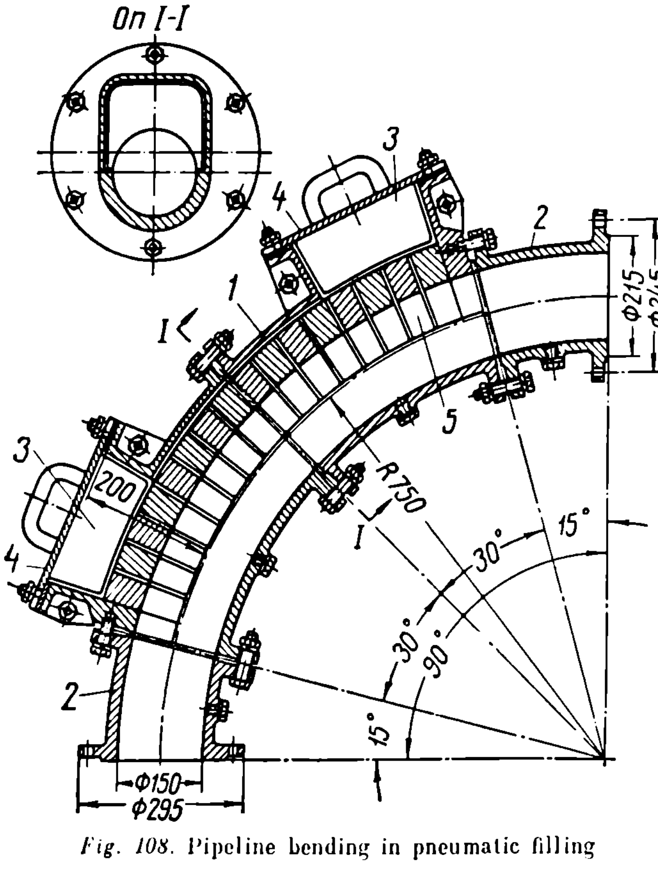



*Fig. 108.* Pipeline bending in pneumatic filling

The advantages of pneumatic fill include: compactness of the packed block (shrinkage not exceeding 20 per cent), simplicity of equipment and ease of its handling. On the other hand, pneumatic filling requires big initial outlays, considerable amounts of electric power, heavier expenses in connection with the wear of pipes and for stow meeting set standards as to its size and content of fines, clay admixtures and moisture level.

The use of air-blast stowing with different methods of mining is described in Chapter XV.

## 8. Hydraulic Fill

As stated before, the term *hydraulic fill* implies the use of some loose or fragmentised rocks, or artificial materials mixed with water and delivered through pipes to a mined-out area to be filled. The mine-fill proper remains in the worked-out space, while the water runs into special collectors and is then pumped out to the ground surface. After it has contracted, the hydraulic fill decreases in volume by only 5-15 per cent and therein lies its main advantage.

Among the disadvantages of hydraulic fill are its high initial cost, difficulties in combining stoping operations with stowing, increased humidity in the mine, the necessity of pumping slime water back to the surface, soiling of underground workings with the silt washed away from the spaces being filled, and the complicated nature of stowing operations in winter time.

The best *material* for hydraulic fill is quartz sand because it readily mixes with water, is easily driven along the pipes and rapidly emits water, which in this case is relatively pure. The resultant fill is very dense. The drawback of the sand used as a mine-fill is that it causes the pipes to wear out rather rapidly.

Clay causes almost no wear, but it can clog the pipes and, besides, does not readily emit water it contains.

In favourable local conditions the following are the materials that can be employed for hydraulic fill: waste rocks from the concentration mills of coal mines (these require much water, and because of their high pyrite content make this water erosive and thus damage pipes); granulated slag (it is cheap, moves easily along the pipes but wears them out considerably and does not make the fill sufficiently compact); boiler cinder (an occasional source).

At the site of their occurrence, loose rocks are excavated by power shovels and delivered to the mine by locomotives or by any other mechanical means in cars provided with adequate facilities for quick automatic unloading. In exceptional cases, when large occurrences of loose rocks are available in the immediate vicinity of a mine (or a borehole) with pipes for the delivery of the fill into underground workings, they can be worked by the hydraulic method. To accomplish this, the loose rock is washed away by powerful jets of water from nozzles or hydraulic giants under a pressure which is 6-7 atm for sand and 15-20 atm for denser clay grounds. In exceptional cases, this is done under still higher pressure. The resultant liquid mixture (*pulp*) goes directy into the mine.

When there are no deposits or piles of loose rocks in the neighbourhood of the mine, crushed hard rocks may be utilised for hydraulic fill, though naturally those which do not require too much mechanical energy to break them. The maximum permissible size of the particles is 60-70 mm. Crushers and sorting screens are used to obtain the fill of proper size.

Good results may frequently be achieved by mixing materials of two different groups. Special tests have proved that quartz sand contracts 5.1 per cent, shales with grain size of 10-25 mm 27.5 per cent and a mixture of 40 per cent of shale and 60 per cent of sand no more than 6-9 per cent. In other words this cheaper mixture is almost equivalent to pure sand by its contraction coefficient. There are also other mixtures that do away with the above-cited disadvantages.

Because of the subsidence of wall rock prior to filling, the abandonment of unstowed workings amidst filled areas, the timbering, etc., only 75-80 per cent of the mined-out area is subject to stowing in hydraulic fill. Since one ton of hard coal with specific weight of 1.25-1.3 in place has a volume of 0.8-0.77 cu m that of the fill needed per ton of coal mined will be around 0.58-0.64 cu m.

One of the main issues to be solved in putting up a unit for hydraulic filling is to determine the *maximum distance* over which the fill can actually be transported along horizontal workings.

This problem is of vital importance for mining in the U.S.S.R. The horizontal travel distance is, first of all, dependent on the head that is built up in the vertical portion of the stowing pipeline underground. The deeper the level of underground mining the higher the head and, consequently, the greater the distance over which the fill can be transported by water along crosscuts and entries. In the U.S.S.R., hydraulic fill can be useful only in mining thick seams. But deposits with thick seams are found exclusively in newly developing areas where, with but few exceptions, the levels mined do not lie deep and the pressure head, therefore, is low. For that reason, in hydraulic stowing, we have to pay particular attention to the maximum possible horizontal travel distance of mine-fill.

Let us discuss this issue in more detail. The pressure under which the mixture of water and fill moves along the pipes is equal to the product of the head, that is, of the vertical distance between the point at which the mixture is discharged on the surface and the given underground level, and the average density of the mixture.

The speed with which the pulp travels in the pipe should not drop below a certain limit if the solid particles are to be driven by water, for otherwise the pipes will immediately become clogged. This critical velocity is conditional upon the size of the material; larger-sized particles require higher speed for their transport by water than the smaller ones.

The critical velocity tends to rise when the pipe runs upgrade, and this, therefore, should be avoided whenever possible. When the fill includes clay shales with medium-sized particles, the critical velocity of the pulp should not be allowed to drop below 3 m/sec. To prevent pipes from choking, the actual velocity of the pulp stream in any given section of the pipeline should be greater than the abovementioned minimum. It is contingent upon the pressure head prevailing at the point where the level portion of the pipeline starts and the friction losses within the part which lies between the starting point and the cross-section under consideration. This loss is proportional to the length of the stream travel and the square of its velocity. Let us assume that the friction losses per unit of length of the pipeline are approximately equal in both its vertical and horizontal portions.

Let us designate by:

$H$—depth of the shaft accommodating the stowing pipeline or, to be more precise, the vertical pressure head in metres;
$L$—total length of the level portion of this pipeline in metres;
$\delta$—average density of the mixture;
$k$—coefficient of resistance to the movement of the mixture in pipes;
$v$—critical velocity of the mixture, m/sec;
$w$—actual velocity of the mixture, m/sec.

Then, in conformity with the statement above, there should be a relation:

$$\delta H \geqslant k(L+H)v^2, \tag{1}$$

whence

$$\frac{L}{H} \leqslant \frac{\delta}{kv^2} - 1. \tag{2}$$

We have seen that actual velocity cannot be below the critical

$$w \geqslant v. \tag{3}$$

Therefore, condition (2) may be rewritten as follows:

$$\frac{L}{H} \leqslant \frac{\delta}{kw^2} - 1. \tag{4}$$

Knowing the numerical values of the three parameters making up the right member of equation (4), it is possible to determine value $L$.

The density of the pulp depends on the volume weight of the fill and the amount of water in a unit of volume of the mixture. The specific weight of clay shale in situ is about 2.3, but its volume weight in lumps amounts to but 1.6 metric tons per cu m. With water consumption being from 1 to 3 cu m per 1 cu m of shale, the density of the mixture will be from 1.24 to 1.53. In a numerical example the last figure is arrived at through the following calculation: The aggregate weight of 1 cu m of lumpy shale and 1 cu m of water is 2.6 tons. The volume of the actual solid body in 1 cu m of lumpy shale is 1.6:2.3=0.7 cu m. Hence, the mixture of 1 cu m of lumpy shale and 1 cu m of water will give us a total volume of 1.7 cu m, whence the density of the mixture will be 2.6 : 1.7=1.53.

As seen above, critical velocity $v$ may be taken at 3 m/sec.

The nature of coefficient of resistance $k$ is very complex. Its value depends upon the diameter of the pipe, the length of individual pipeline sections, the accuracy of their alignment, properties and size of fill particles and the percentage of water, and generally speaking, is inversely related to the latter. More often than not, this coefficient ranges from 0.01 to 0.03, but in unfavourable conditions it may be much greater.

If, for instance, we take $\delta=1.24$, $w=3$, $k=0.01$, then, according to formula (4), the resultant relation will be $\frac{L}{H}=12.8$. With $\delta=1.53$, $w=3$ and $k=0.03$ $\frac{L}{H}=4.6$.

For hydraulic fill with crushed rocks in the conditions prevailing in the Kuznetsk coal fields, it is recommended to assume that relation $L:H$ ranges from 5 to 8.

To avoid pipe clogging, the fill should be mixed thoroughly with water.

The *mixing units* used for the preparation of pulp are set up either on the surface or underground. Surface location has the advantage of permitting mixing operations in daylight; there is sufficient room for accommodating the equipment and a high pressure head can be built up in the pipes, thus facilitating the transport of the mine-fill. On the other hand, the water escaping from the fill must be pumped back to the surface. Furthermore, the pipeline wears out more intensively, particularly in the region of the lower main bend, and pipe clogging can-

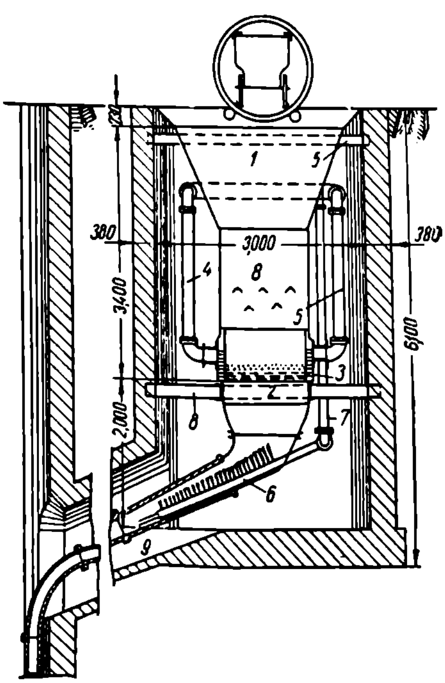



*Fig. 109.* Mixing unit with a funnel

not be ruled out. These disadvantages are felt all the more as the mine deepens.

Mixing units are mainly of two types: *funnels* (Fig. 109) or *spouts* (troughs) (Fig. 110) communicating with mine-fill *storage space.*

Mixing funnel *1* has grating *2* at the bottom. Over the grating the funnel is surrounded with circular tank *3* into which water is fed through pipes *4* and *5*. The wall of the funnel has many perforations through which the supplied fill can be intensively sprayed with water. The wetted fill passes through the grating and is once more watered in inferior sloping portion of the funnel *6* by streams escaping from perforations in the pipe *7* and, finally, by a jet from the end of the pipe. At a certain height over the grating, inside the

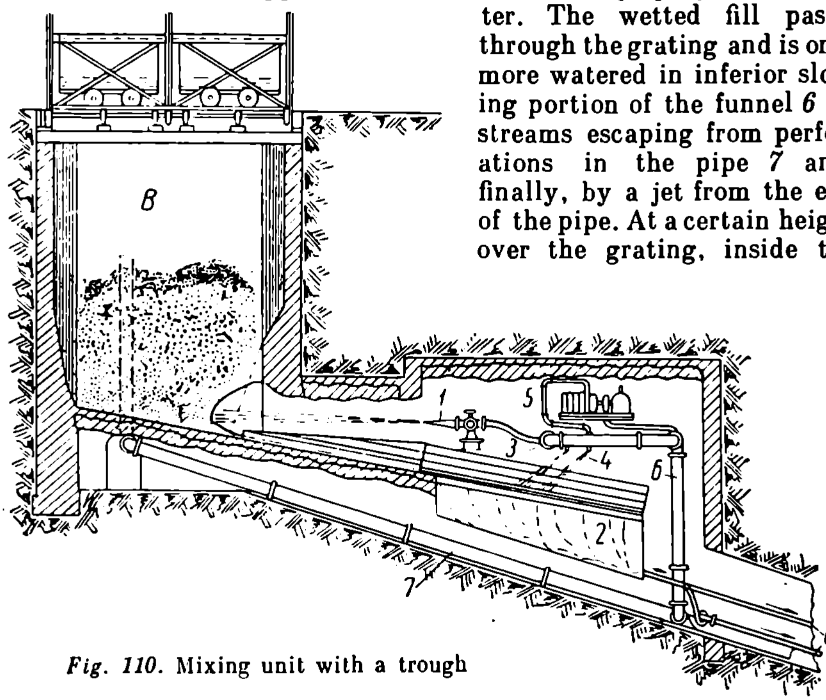



*Fig. 110.* Mixing unit with a trough

funnel, several angle bars *8* are fixed, their ribs directed upwards so as to permit breaking up oversized lumps that may get into funnel with the fill. The funnel itself is supported by solid beams. The entire installation is set up below the ground level and is connected with pipes running in the shaft through inclined opening *9*. To ensure adequate stowing, the fill should be supplied intensively and uniformly, and that is difficult to realise by using mine cars to deliver it to the funnel. For this conveyer feeding is more suitable.

Stowing operations are even better ensured when the mixing unit communicates directly with the fill storage room (Fig. 110). Because of their higher efficiency, installations of this type are used quite frequently, The fill storage room is located below the ground surface, this simplifying transport and unloading facilities. The fill is mixed with water not in the funnel, but in the trough. Bin *B* has a capacity of 220 cu m (generally, this capacity may reach as much as 500 cu m). The fill is washed out by jets of water emitted under a pressure of 4 atm by two hydraulic giants *1*, then mixes with water and flows down along an inclined plate towards grating *2*. The fines fall through the grating, while oversize lumps of hard rock are removed and sent to a crusher. The chunks of clay are broken up by two additional streams *3* and *4*, thrown out by special pump *5* under a pressure of 12 atm. This pump gets its water supply from branching *6* of general pressure pipeline *7*. To safeguard it from the destructive effect of water streams emerging from pressure nozzles the floor of the bin is covered with thick plates or hard-burnt bricks. The fill mixture is delivered to the mine via two pipelines *8*.

The amount of water required to obtain a sufficiently fluid mixture is determined in the fill storage room automatically, all by itself. In the case of pure sand, the mixture runs down the inclined floor when the ratio is 1:1. Larger-sized crushed material, on the other hand, is harder to be driven by water and to make such mixture fluid the volume of water must be increased to a ratio of 2 : 1, or 2.5 : 1 and sometimes even more. To prevent clogging, pure water is let into the stowing pipes for 2-3 minutes before the mine-fill is fed. Thereafter the water stream is directed at the piles of sand (or crushed material), but in a manner preventing the amount of the material entrained being too great. Following this, the density of the mixture is brought to the level indicated above.

All the production sections of the mine are linked with the fill storage room by telephone and call bells. Operations in the storage room are thus started and ended on instructions received from below. The signalling system is being automated.

Shortly before the operation of delivering mine-fill is completed, the pipes are flushed with pure water for 2-3 minutes.

Fig. 111 depicts the arrangement used for the preparation of stowing pulp at one of the mines in the Kuznetsk coal fields. From the bin the mine-fill (crushed rock) is fed to mixing platform *2* and then to trough *3*. The water is supplied from pressure nozzle *1*. The mixing platform and the trough are hinged together and suspended by rod *5* so as to make it possible to change the angle of their inclination with the view to controlling pulp density. Inside main receiving hopper *4* there is an additional hopper—*6*. When work proceeds normally, the pulp is fed into the mine stowing pipeline through hopper *6*. When it becomes clogged, the pulp overflows its edges into the annular space between the two hoppers, thus ensuring continuous supply of the mine-fill.

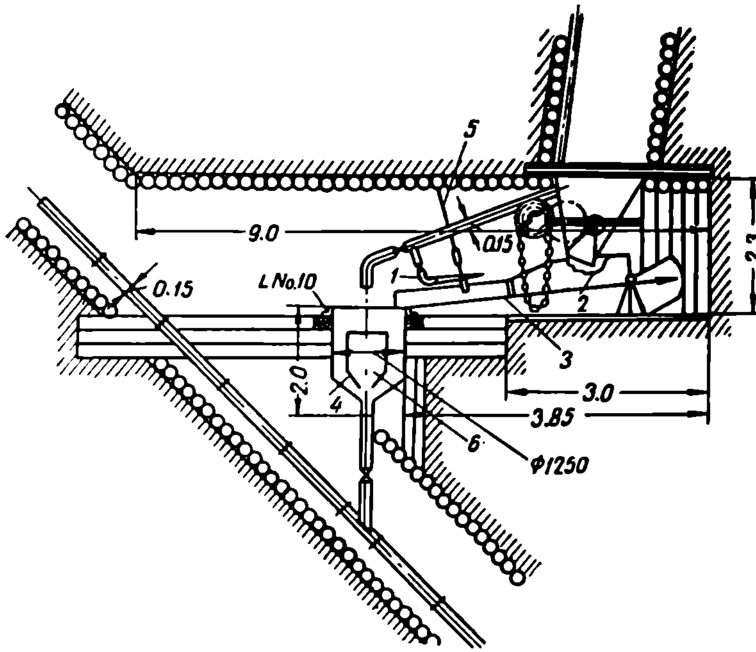



*Fig. 111.* Mixing plant at one of the mines in the Kuznetsk coal fields

Fig. 112 depicts an underground mixing plant at one of deep levels of the mine. Dry mine-fill is let down along vertical pipe *1* until it reaches "cushion" *2* made of the same material (to cushion the impact), passes along inclined opening *3* to underground bin *4*, set in concrete, and thence, via a chute, it may be discharged into hopper *5*. The latter is supplied with water and, after it has mixed with it, the fill is sent through pipes *6* to its destination. An underground mixing plant should be installed over the production level, at a height sufficient to guarantee the necessary head. The same drawing shows the arrangement of a plant for crushing hard mine-fill—waste rock from dumps, in this instance. From mine cars in a tipper the rock gets onto inclined grizzly *7*, which lets the fines through, while oversize lumps go to crusher *8*.

The mine-fill is delivered to underground workings through pipes laid in shafts. Less often, special boreholes with a large diameter (0.8-1.5 metres) are driven for this purpose.

Pulp is delivered along main lines by seamless steel *pipes* with a diameter of 150-200 mm, 5-6 metres long and with walls not less than 8 mm thick. Circulation branch pipes in the stopes are 2-3 metres long and their walls are about 3 mm thick. These dimensions facilitate their handling.

Since the *wear of pipes* constitutes a very substantial item of expenditure in hydraulic fill, this aspect of operations should be given particular attention. The various parts of the pipeline by far do not wear out to the same extent. The following is the descending order of importance: main bending (that is, the one connecting the pipeline in the shaft with other underground pipes), bendings in inclined pipes, in level and slightly sloping ones and, finally, in vertical pipes. Inasmuch as one side of the pipe is chiefly liable to wear out—for example, the lower one in horizontal workings—the pipes in use should be turned round by 120° as they wear out.

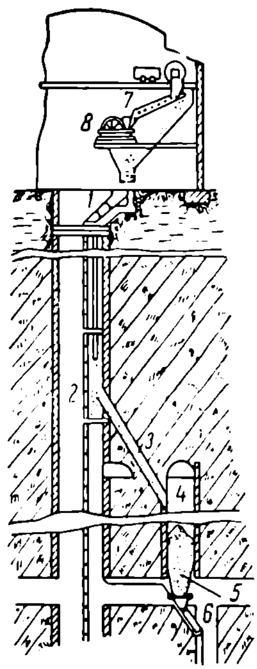


*Fig. 112.* Underground arrangement of mixing plants with hydraulic filling

In sections wearing out most, the pipes employed are usually made of variable thickness, the wall on the side liable most to wear being the thickest. Such pipes, however, are rather costly. Very often pipes are lined with pig-iron, hard steel, ceramics, etc. (Fig. 113).

To reduce frictional resistance and wear of horizontal and inclined pipes, they are sometimes made to have an oval cross-section.

As stated above, to prevent clogging the velocity of the pulp flow should not be allowed to drop below 3-3.5 m/sec. The maximum permissible speed of the flow is 7 m/sec.

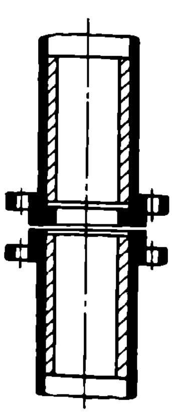


*Fig. 113.* Ceramic pipe lining

Elimination of chokes in main pipelines requires arrangement of inspection holes in the form of tee-pieces with screw caps every 50-80 metres. Moreover, there should be such inspection holes in the vicinity of all bendings.

To preclude choking, the bendings should have a fairly large radius. At bendings, the pipes should be provided with holes permitting them to be cleaned directly or by pressurised water jet (Fig. 114).

In order to reduce the wear of stowing pipelines at the bendings when sand is used for filling—in addition to using steel pipes, thickening their walls or lining them—they are sometimes provided with ribs (Fig. 114) which hold back the sand and thus protect the wall at the bending from excessive wear. To deliver the fill in the desired direction, special valves are mounted at pipeline bendings. The design of one is illustrated in Fig. 115. The valve axle is extended outside the pipe and handle *1* is pressed on its end. In the right or left position, this handle is fixed by holder *2* which is tightly bolted to lugs *3* of the valve casing. By changing the position of valve flap *4* the flow of the mine-fill can be directed either into pipe *5* or *6*. The valve casing has inspection hole *7*, provided with a cover.

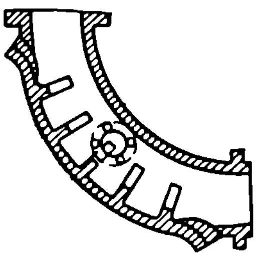


*Fig. 114.* A pipe with inside ribs at the bending

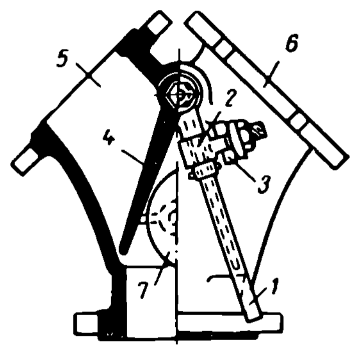



*Fig. 115.* Flap valve at a pipeline branching

The details of hydraulic fill flushing, related to mining methods proper, are given in the text devoted to the description of these methods.

Here we shall dwell briefly upon *clearing* and *lifting* of water that escapes from the fill during the stowing of mined-out areas. This water at times is clear (when the fill is made of sand) and at others muddy (when the fill is clayish). To clear the water, it is made to settle in tanks which provide for the maximum possible sedimentation of mineral particles (decelerated flow, bendings, fascine spacings for filtration, etc.), or else it is made to flow through abandoned mined-out areas and rock amassments where the slime settles down. The dirt accumulated in tanks must be removed from time to time and this is done by special pumps (for example, air-operated ones).

Clarified water may be brought to the surface by main mine-drainage pumps, but to avoid their excessive wear this is done only when the water is completely cleared underground. Muddy water is lifted by special pumps, preferably rotary ones, their intense and rapid wear being compensated for by a more convenient arrangement of settling tanks for muddy water on the mine surface,

## 9. Comparison of Different Mine-Fill Types and the Spheres of Their Application

Despite all its simplicity, *hand* packing has one major disadvantage—it is a highly labour-consuming operation. Therefore, its use is restricted to exceptional cases in which work is done on a small scale; for example, in packing diagonal workings, strip packing of mined-out spaces in thin gently sloping beds, etc. But even these operations should be mechanised, for instance, with the aid of conveyers and slushers.

The advantages of the fill *by gravity* are: 1) ease of operation; 2) possibility of using mine-fill of variable size—with lumps up to 120mm and over; 3) high efficiency of stowing operations. On the other hand, this type of fill has also considerable shortcomings, such as: 1) necessity of providing special transport facilities for bringing the mine-fill to the sloping area, since it is moved by gravity only within the stopes proper; 2) difficulties in delivering moist or clayish ground by conveyers or in mine cars to the spot on the surface whence it is brought down into the mine; 3) need of supplementary mechanical facilities to secure tight stowing in the upper portions of stopes. Stowing by gravity can be employed in stopes with a moderate or high dip.

The positive aspects of *mechanical* stowing include: 1) simplicity of equipment; 2) sufficient compactness of the fill; 3) low power consumption; 4) ease of stowing the upper portions of workings. On the other hand, among the drawbacks of mechanical filling are: 1) the fact that it is only the stowing of mine-fill proper that is actually mechanised, while the material itself must be delivered to a stowing machine with the aid of other transport facilities; 2) difficulty of manipulating stowing machines in the stopes; 3) heavy wear and tear of belts; 4) excessive dust formation requiring special measures to combat it. Therefore, it is advisable to use these machines principally as ancillary equipment for stowing the fill in the space immediately under the roofs of workings.

Among the advantages of *pneumatic* fill should be listed: 1) extreme compactness of the mine-fill block; 2) simplicity of stowing operations in stopes; 3) ease of fill transportation through pipes within the range of the mining field. The shortcomings of this method include: 1) necessity of maintaining heavy-duty air and power equipment; 2) high cost of machinery; 3) high power consumption rate; 4) increased wear and tear of machines and pipes; 5) need of specially made mine workings to accommodate air-operated machines; 6) possible "cloggings" of pipelines; 7) dust formation in stopes and, hence, the necessity of supplying water to reduce it.

Considering the technical features specific to air-blast stowing equipment, the field covered by pneumatic fill is manifold indeed, but it is predominantly used with mining methods for the level arrangement of working stopes.

*Hydraulic* or *float* fill has the following major advantages: 1) high degree of compactness; 2) automatic transportation of the fill by water jets from the ground surface right to the face of the stope; 3) possibility of achieving a high degree of stowing efficiency; 4) simplicity of stowing operations. The disadvantages of the method include: 1) introduction of water into stopes and other mine workings; 2) impossibility of utilising any materials other than sand and small-sized crushed rock; 3) difficulty of arranging seals or bulkheads; 4) heavy consumption of water, its clearing and back pumping; 5) necessity of clearing workings of slime; 6) difficulties arising from the organisation of proper water supply for stowing operations in winter.

The principal spheres for the application of hydraulic stowing are thick beds with sand deposits occurring nearby. If there are none, crushed rock may be used. In stopes with moderate and high dip the stowing of hydraulic fill tends to produce excessive pressure against bulkheads.

The use of various types of fill is now of particular importance in mining steeply dipping coal beds in the Kuznetsk fields.

# COAL DEPOSITS

CHAPTER IX

## CHOICE OF MINING METHODS AND MODES OF EXTRACTION

Section 2 of Chapter VII briefly enumerated the factors influencing the choice of mining methods in working deposits of different minerals. This chapter is devoted to a detailed analysis of these factors when applied to mining of coal deposits.

### 1. Shape of Deposits

Mineral coals occur in the earth's crust in the shape of beds (seams) or sheetlike deposits.

In its ideal form a bed looks like a tabular body of uniform thickness, which is insignificant compared to its two other dimensions. In nature beds of such ideal form are nonexistent—due to genetic and tectonic causes, the thickness of a bed is subject to variations (bulgings, attenuations, peterings out), its continuity may be

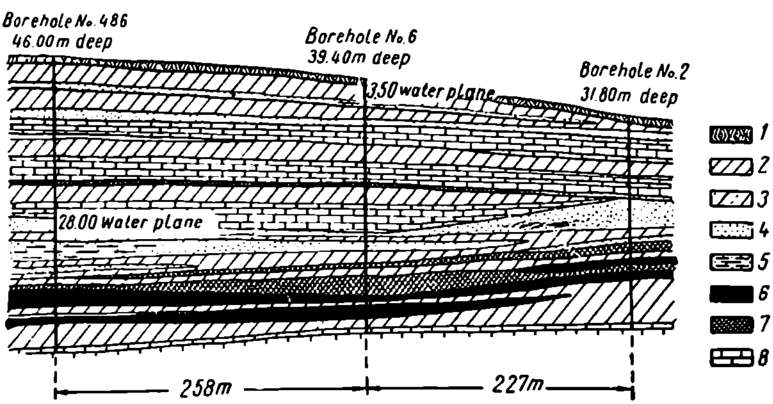



*Fig. 116.* Typical occurrence of coal seams in the Moscow coal fields

1—chernozem; 2—clay; 3—sandy clay; 4—sand; 5—aquiferous sand; 6—coal; 7—blossom; 8—limestone

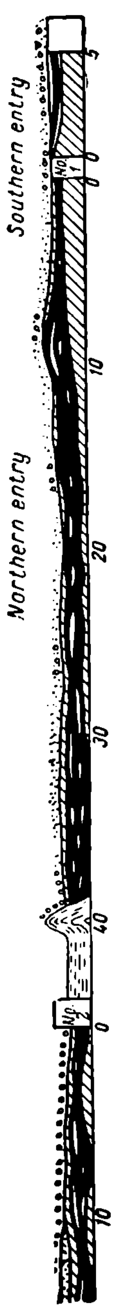



*Fig. 117.* An example of abrupt changes in the thickness and structure of a coal seam (Kamensk deposit in the Urals)

broken, the lines of the strike and angles of dip are apt to change over short distances.

Of quite regular shape are the beds in many districts of the Donets coal fields. The Moscow coal basin is distinguished by sheetlike deposits (Fig. 116). One example of strongly pronounced variability is some of the coal deposits on the eastern slope of the Urals (Fig. 117). A picture of extremely complex shape and structure is presented by the rich Korkino lignite deposits lying in the Urals south of Chelyabinsk (Fig. 118).

Frequent variations of thickness and other elements distinguishing the occurrence of seams, such as roughness of the floor and roof, swellings, attenuations and peterings out of beds, faults and shifts, even of the slightest amplitude, tend to complicate the mining of a deposit and should be taken into account in planning and elaborating a suitable mining method.

## 2. Thickness of Seams

The thickness of beds is a factor of paramount importance in the selection of a mining method, for it determines the mode of extraction, the nature of wall-rock cavings over mined-out spaces, the necessity of using back fill or the possibility of doing without it. In thicker beds, the worked-out rooms are higher and the displacement of rocks overlying them progresses with greater intensity.

If *full-seam extraction* were practised in the working stope of a thick bed, the excessive height of production faces and the large area of coal exposed during extraction would make stoping operations both inconvenient and unsafe. Therefore, in working beds of considerable thickness the frequent practice is to employ so-called *slicing* methods, that is, work a thick bed by extracting individual slices 2-3 metres thick.

In thick beds, particularly those with a steep pitch, filling of mined-out areas quite often proves indispensable.

By their thickness coal seams are subdivided into four groups: 1) *very thin*—up to 0.5 metres; 2) *thin*—from 0.5 to 1.3 metres; 3) of *medium thick-*

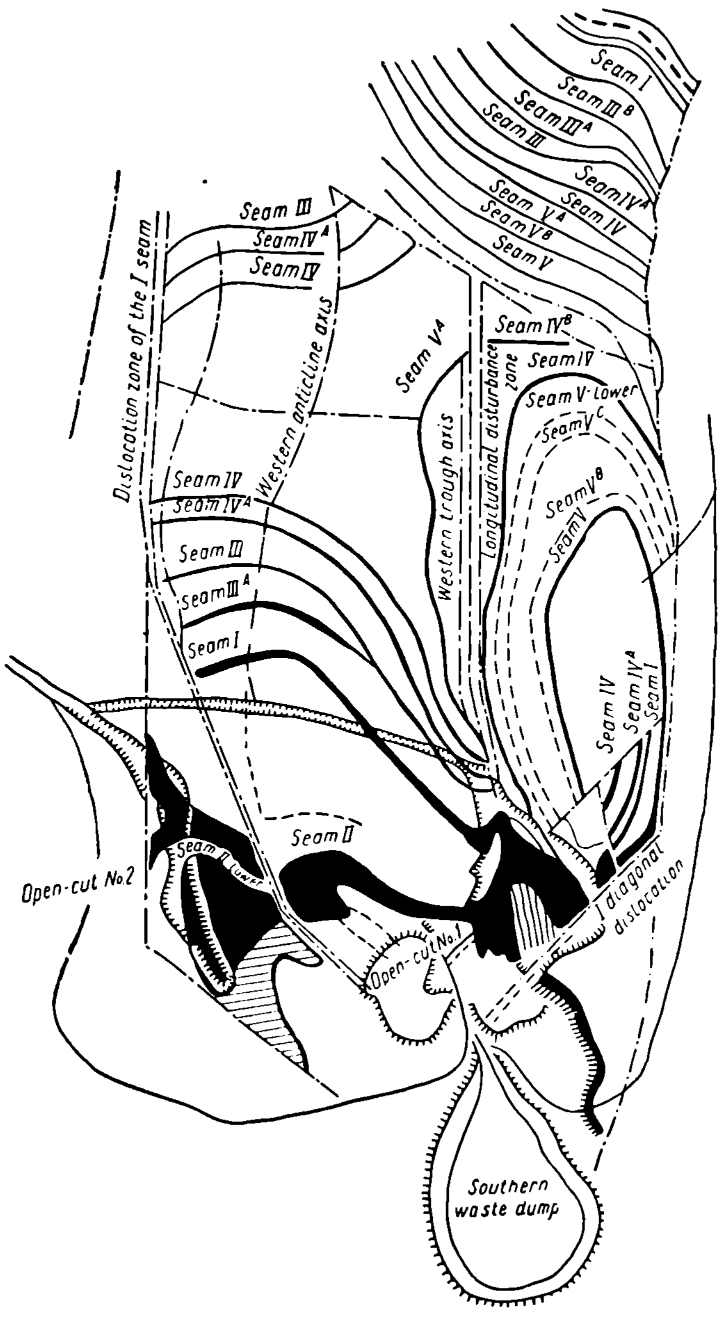



*Fig. 118.* A lignite deposit of an extremely complex shape (Korkino in the Urals)

*ness*—from 1.3 to 3.5 metres and 4) *thick* or *high seams* — over 3.5 metres.

Such classification is usually motivated by the following, rather conventional, considerations (Fig. 119). The group of very thin and thin seams includes those whose thickness is inferior to the usual height of development workings, suitable to human stature (*1*). To the category of seams with medium thickness belong those whose thickness is approximately the same as the height of the above-named workings (*2*). The thickness of beds of the third group exceeds, often quite appreciably, the height of development openings (*3*). The conventional nature of this grouping consists in that, firstly, the height of development workings does not represent a definitely established magnitude and, secondly, the position of a bed in relation to the contours of the working may differ in many ways, depending upon

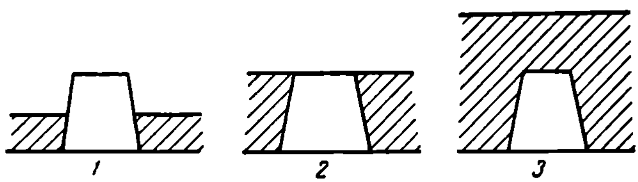



*Fig. 119.* Grouping of seams according to thickness

the angle of dip. Nevertheless, the above-mentioned classification of coal seams according to their thickness—their subdivision into very thin, thin, medium and thick—is very convenient for practical purposes.

Coal beds whose natural properties make them suitable for extraction are called *pay beds* or *payable seams.*

The *minimal* thickness of pay beds is ordinarily assumed to be around 0.4-0.5 metre, although there have been instances of even thinner seams—0.3 metre thick—being worked quite successfully. If a coal measure includes very thin and thicker seams, the first should also be extracted, though their mining entails somewhat greater expenditure. If this is not done, coal reserves in very thin seams are irretrievably lost for the national economy.

Coal seams not fit for extraction because of their thinness are called *coal sheds.*

The thickness of beds encountered in diverse coal fields varies widely.

In the Donets coal fields thin beds predominate. Seams up to 1.5-2 metres thick are an exception to the rule here.

In the Moscow basin the usual thickness of beds is 1.5-3 metres and it is only in some places that it exceeds this figure.

In the Kizel coal fields of the Urals there are thin beds and beds about 4-6 metres thick. Brown coal deposits occurring on the eastern slope of the Ural Mountains (Bogoslovsk and Korkino are the principal ones) include beds of immense thickness, reaching scores of metres (the maximum thickness of the lignite deposit at Korkino comes to as much as 160 metres. This is the thickest known coal deposit in the world).

The coal-bearing formations of the Karaganda basin include thin seams, beds of moderate thickness and thick ones—to 7-9 metres (the Verkhnaya Marianna seam).

Very rich in coal beds is the Kuznetsk basin. The thickest beds are in the southwestern part of the basin—in the Prokopyevsk-Kiselyovsk area, where along with numerous thin and moderately thick seams one comes across beds as much as 15-16 metres thick (the Moshchny seam) and in some places even thicker.

In the Cheremkhovo area (west of Irkutsk) coal is won from the Glavny seam, which is about 7-9 metres thick.

Thick beds are also met with in many other basins and areas of the U.S.S.R. (Tkvarcheli and Tkvibuli occurrences in Georgia; Angren, Sulyukta, Shurab, Kizil-Kiya and others in Central Asia; lignite deposits on the right bank of the Dnieper River in the Ukraine; deposits in Bashkiria and Kazakhstan; in the Far East and in other parts).

To certain depths from the ground surface thick beds are mined by the open-cut method.

## 3. Angle of Dip

As said above, coal seams are classified by their angle of dip into *gently sloping* (0-25°), *inclined* (25-45°) and *steeply pitching* (45-90°).

The pitch of a coal seam is one of the major factors taken into account in selecting a mining method.

While in a gently sloping bed coal lumps broken in the slope, or fallen rock blocks, remain on the spot, in steeply pitching beds they roll down the dip. To protect workers and face timbering from being hit by falling objects, the mining method used must have corresponding structural features. In steeply pitching seams, in contrast to gently sloping ones, it is not only the roof of the bed that can collapse; its bottom may start creeping too. While in gently sloping beds extracted coal is transported mechanically (chiefly by conveyer), in the faces of high dipping seams it moves by gravity.

Abrupt changes in the dip of seams seriously complicate their working.

## 4. Structure of Seams

The structure of a coal seam is an important factor in the choice of methods and the sequence of extraction.

There are *pure* seams, that is, seams *without gangue bands* (Fig. 120) and seams *with intercalations*, also known as *multiple* or *composite* seams (Fig. 121).

But even in pure or homogeneous seams the properties of coal (or any other useful mineral) may vary. Portions of a seam possessing special features are called *benches*. For example, Fig. 120 shows a pure or homogeneous seam consisting of two benches—*a* and *b*.

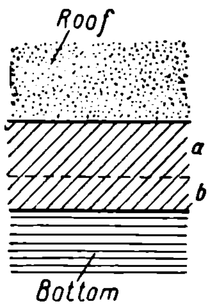


*Fig. 120.* Homogeneous ("pure") seam

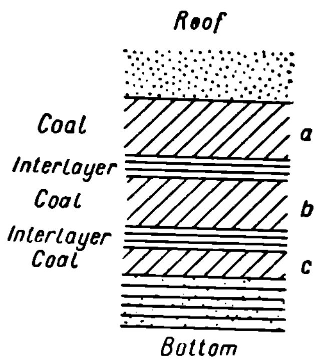


*Fig. 121.* Intercalated seam

Portions of a composite seam separated by gangue bands are also called benches. For instance, the seam shown in Fig. 121 has two gangue bands and three coal benches—top *a*, medium *b* and bottom *c*.

A specimen of a high coal seam of very complex structure (Verkhnaya Marianna in the Karaganda coal fields) is depicted in Fig. 122.

The presence of gangue bands complicates the extraction of seams. If the gangue from interlayers is allowed to mix with coal, this entails a very adverse increase in ash content. But separation of interlayers less than 5 cm thick is practically impossible during the stoping of coal in a seam. When coal is loaded by hand, the hewers can pick up waste from thicker bands and throw it into worked-out spaces, but this lowers their efficiency. Mechanical loading of coal in the faces makes separation of waste from gangue bands simply impossible. In such cases, the waste is partially picked out in the mine while being transported by conveyers. But, in general, mines working seams with gangue bands and possessing no facilities to separate the waste from coal at the face should have their own concentrating mills. If the coal mined is used as fuel in power plants and burnt in coal-dust furnaces, it may be profitable (which must be confirmed by technical and economic calculations) to use raw crushed run-of-mine coal.

High-pitching seams present particular difficulties for the separation of gangue bands from the bulk of coal.

In mining high seams of composite structure which are divided into individual inclined layers (lying parallel to the bedding plane) efforts should be made to use individual intercalations or their groups for separating these layers.

The waste contained in gangue bands separated from coal when it is broken is usually thrown into mined-out areas. If the volume of waste is not too small, it can be employed for packing. Waste containing carbonaceous matter and liable to spontaneous combustion should not be kept in mined-out areas.


*Fig. 122.* A seam of composite structure (Verkhnaya Marianna in Karaganda)

*I* — clay shale; *II*—coal; *III*—ampelite

## 5. Quality Characteristics of Coal

Mineral coal comprises organic and mineral constituents.

The *organic* matter of coal is a complex and multiform aggregate which includes chemical compounds of carbon, hydrogen, oxygen, nitrogen and sulphur. Proportional content of these elements in coal is established by *elementary analysis*.

For instance, the following elementary composition of organic matter in per cent is typical for Donets basin coal (Table 3).

The organic mass of the Moscow basin (lignite) coal includes: 71-75 per cent of carbon, 3.8-5.0 per cent of hydrogen, 18.5-23.5 per cent of oxygen and 1.4-2.0 per cent of nitrogen.

Besides elementary analysis the commercial quality of coal is also determined by proximate, plastometric and petrographic analyses. In addition to this, coal is examined for its ability to undergo benefication and is put to dry distillation, fusion and mechanical tests.

1. *Proximate analysis* of coal is of utmost commercial importance. It helps to determine the content of moisture ($W$), ash ($A$),

*Table 3*

**Composition of Organic Matter in Donets Coal**

| Elements | Commercial brands of coal | | | | | | |
|---|---|---|---|---|---|---|---|
| | Д Long-flame | Г Gas coal | ПЖ Steam-fat coal | К Coking coal | ПС Steam-baking coal | Т Lean coal | А Anthracite |
| Carbon . . . . | 76-86 | 78-89 | 84-90 | 87-92 | 89-94 | 90-95 | 95-97.5 |
| Hydrogen . . . | 5-6 | 4.5-5.5 | 4-5.4 | 4.5-2 | 3.8-4.9 | 3.4-4.4 | 1.2-2.7 |
| Oxygen . . . . | 10-17.5 | 6.8-16 | 5-10.5 | 3-8 | 2-5 | 1.6-4.5 | — |
| Nitrogen . . . | 1.8 | 1.7 | 1.7 | 1.5 | 1.5 | 1.2 | 1.5-3.5 |

sulphur ($S$), phosphorus ($P$), volatile matter ($V$) and calorific value ($Q$).

In estimating the moisture content (State Standards 147-41-III-VI) distinction is made between:

*moisture in effective fuel* ($W_f$) or, which is the same, *total moisture* in coal ($W_t$), this representing all the moisture contained in coal, except for crystal water, which is mainly part of the mineral (aluminium silicate) portion of coal and is not determined by proximate analysis;

*external* or *surface moisture* ($W_e$), dependent on the extent to which the seam is exposed to the action of external water sources which can be relatively easily eliminated by drying in the open; so-called *internal* or *hygroscopic moisture* ($W_h$), seated in the pores and capillaries of coal and dependent on its chemical nature. This moisture can be eliminated only by desiccating coal at a temperature not below 102-105°. In the laboratory, moisture is determined by an analytical test of coal and designated as $W^a$ (analytical moisture).

Surface moisture content $W_e$ is determined by the loss of weight of a coal sample after it has been dried down to its constant weight in the open at room temperature or in a drying cabinet heated to 70° (±5°).

Analytical moisture content $W^a$ is estimated by the loss of weight of a laboratory sample after it has been desiccated to the point of constant (dry) weight in a drying cabinet heated to 102-105°.

Total moisture $W_t$ is found either by calculation or by directly drying the whole of the batch to the point of constant weight in a drying cabinet heated to 102-105°, and by consequent determination of the weight lost. To estimate effective fuel moisture, a special sample is collected and sent to the laboratory in a hermetically sealed metal container.

Coal *ash* is the unburnt mineral residue remaining after the combustion of coal.

Three types of coal ash are distinguished in analytical work: ash content of run-of-mine coal ($A^r$), ash content of an analytical (laboratory) test sample ($A^a$) and ash content of an absolutely dry coal mass ($A^d$).

In the laboratory, ash content in an analytical test sample is determined by burning a coal batch in a small open melting pot, placed in a muffle heated to 800° (±25°) with access of air.

Ash content in effective fuel and the air-dry coal mass is found by calculation according to the following formulas:

$$A^r = A^a \frac{100-W_f}{100-W^a};$$

$$A^d = A^a \frac{100}{100-W^a}.$$

Since mineral substances forming ash play no part in the combustion of coal, ash is a worthless and harmful *ballast*, like moisture. For this reason seams should be worked in a manner of minimising the chances of coal at the face becoming diluted with waste from the roof and floor of the bed. To reduce its ash content, mined coal undergoes benefication or concentration. Low ash content is of particular importance for preparation of metallurgical coke.

In the burning of coal in boiler furnaces or in coke producers a significant role is played by the *fusibility* of its ash. This may be a source of considerable inconvenience and difficulties. The fusing point of ash depends mainly on its chemical composition.

*Table 4*

Marking of Donets Coal

| Commercial marking of coal | Д Long-flame coal | Г Gas coal | ПЖ Steam-fat coal | К Coking coal | ПС Steam-baking coal | Т Lean coal |
|---|---|---|---|---|---|---|
| Volatile matter per combustible mass in per cent | Over 42 | 44-35 | 35-26 | 26-18 | 18-12 | Below 12 |
| Type of coke button | Non-baked, powdery or agglutinated | Baked, fused, sometimes swollen (friable) | Baked, fused, solid or moderately solid | Baked, fused, solid or moderately solid | Baked or fused, from solid to moderately solid | Powdery or agglutinated |

Increased silica and alumina content tends to raise this point, while that of ferrum, calcium and magnesium compounds lowers it. Ash is classified into groups of fusible (below 1,200°), of medium fusibility (1,200-1,300°), low fusibility (1,300-1,500°), very low fusibility (1,500-1,650°) and refractory (above 1,650°).

The increased fusing point of ash in furnaces may be achieved either by mixing various grades of coal or by its concentration.

Coal contains three types of *sulphur*: *pyrite*, *sulphate* and *organic*.

Pyrite sulphur is found in coal in finely disseminated state, or else in the form of a slight streak between bedding planes, thin intercalations or individual inclusions(concretions).Pyrite sulphur and organic sulphur come under the heading of *volatile* modification: in burning, they become transformed into sulphur dioxide and, on coming into contact with moisture, form sulphuric and sulphurous acids which are capable of corroding metallic objects. They are, therefore, also regarded as *harmful*.

Sulphate transforms into coal ash. Much of organic and a slight portion of pyritic sulphur transforms into coke and impairs its quality.

Usually it is not individual types of sulphur in the coal that are determined, but its aggregate amount—total sulphur $S_{tot}$.

The yield of volatile matter from coal is estimated by an analytical sample test in accordance with existing standards (State Standard 147-41-VIII).

Dry distillation or heating of solid fuel in the absence of air results in desintegration of coal and is attended by the evolution of a number of fumes and gaseous products—the so-called *volatile* matter, on the one hand, and by the formation of solid residue—*coke button*, on the other.

The amount and elementary composition of the volatile matter evolved and the external appearance of coke button characterise the nature of coal and its commercial value. The aspect of coke button is illustrative of the degree of *caking* or *baking* capacity of coal and, consequently, of its suitability for coking industry. Therefore, the yield of volatile matter and the nature of coke button are regarded as characteristic signs in the marking of mineral coal.

For Donets coal this marking is shown in Table 4 and for Kuznetsk coal in Table 5.

At Karaganda Brand K includes coal yielding from 24 to 32 per cent of volatile matter. In the Kizel coal fields the group of fat coal (Brand Ж) refers to that with the proportion of volatile matter ranging from 36 to 43 per cent, etc.

To estimate the proportion of volatile matter, a coal batch is heated in a porcelain crucible with the cover carefully ground in, placed in a muffle furnace at 850° (± 20°). The loss of weight in per cent of

*Table 5*

**Marking of Kuznetsk Coal**

| Coal brands | Г Gas coal | ГМ Gas, low fusible coal | Ж Fat coal | КЖ Coking, fat coal | К Coking coal | КО Coking, meagre coal | ПУ Adjuvant fat coal | ПТ Adjuvant lean coal |
|---|---|---|---|---|---|---|---|---|
| Volatile matter in per cent | 43-37 | 35-28 | 37-24 | 34-25 | 25-18 | 22-16 | 28-22 | 17-13 |

the coal batch, minus analytical moisture $W^a$, represents $V^a$—the proportion of the volatile matter sought.

The *calorific value* of coal (State Standard 147-41-XI) characterises the amount of thermal energy contained in it. Its numerical value depends chiefly on the content in coal of combustible components—*carbon* and *nitrogen.*

The availability of other elements in coal—such as oxygen, nitrogen, and moisture and ash—tends to reduce its thermal value.

Determination of the heating value of coal by an analytical test in a laboratory is effected in a hermetically sealed steel vessel (*calorimetric bomb*) in which a weighed quantity of coal is burnt in compressed oxygen. Subsequently, the heat generated in the bomb ($Q^a_b$) is estimated.

Correctives for acids (sulphuric and nitric) formed in the bomb give the *gross* calorific value of coal in the bomb ($Q^a_{gb}$). This includes the latent heat of evaporation of effective moisture and water formed by coal hydrogen. However, in industrial installations (for example, in boiler furnaces) these types of moisture escape into the atmosphere in the form of vapour and do not impart their latent heat of evaporation to the boilers.

Correctives for this value give the *net* calorific value of coal in a bomb ($Q^a_{nb}$).

By their evaluation both modifications of the heating value—gross and net—make it possible to arrive at the calorific value of *effective* fuel, air-dry coal and its organic substance.

Of the greatest practical importance is the *net* calorific value of effective fuel $Q^e_n$, which determines the suitability of given coal for any specific use.

Table 6 gives the figures of the gross calorific value of combustible substance $Q^c_b$ and those of the net calorific values of effective fuel $Q^e_n$ for some brands of mineral coal and lignites in the U.S.S.R.

The heat value of coal in other coal fields of the Soviet Union is approximately the same.

*Table 6*

**Calorific Value of Some Types of Coal, in cal**

| Brands of coal | Mineral coal | | | | Lignites | | |
|---|---|---|---|---|---|---|---|
| | Donets | | Kuznetsk | | Deposits | $Q_b^c$ | $Q_n^e$ |
| | $Q_b^c$ | $Q_n^e$ | $Q_b^c$ | $Q_n^e$ | | | |
| Д | 7,500 | 5,250 | 7,700 | 6,400 | Moscow basin | 6,600 | 2,800 |
| Г | 8,150 | 5,700 | 8,200 | 6,850 | Northern Urals (Karpinsk) | 6,250 | 2,800 |
| ПЖ | 8,500 | 6,500 | 8,550 | 7,000 | | | |
| К | 8,500 | 6,500 | 8,500 | 7,050 | Southern Urals (Chelyabinsk) | 6,900 | 3,900 |
| $К_2$ | — | — | 8,500 | 7,050 | | | |
| ПС | 8,500 | 6,600 | 8;550 | 6,900 | Ukraine (Aleksandria) | 6,600 | 2,000 |
| СС | — | — | 8,300 | 6,800 | | | |
| Т | 8,500 | 6,750 | 8,450 | 6,650 | | | |

2. *Plastometric analysis of coal.* The suitability of coal for the preparation of metallurgical coke is regarded as one of its valuable properties. Not all coal is capable of producing metallurgical coke of good quality. Therefore, in most instances it is necessary to resort to mixing various brands of coal (preparation of coal *stocks*) whose coking properties are close to those of natural coking coal.

Preliminary determinations of the coking properties of any coal—individually or in a mixture—is of a considerable practical importance.

Yet until quite recently the coking properties of coal were determined by the amount of volatile matter it yielded. This procedure, however, did not prove quite satisfactory, since coal yielding the same amount of volatile matter may nevertheless have different coking properties, this depending on the chemical composition of volatile matter.

Coking properties of coal may also be determined by the *plastometric method* proposed by L. Sapozhnikov and now widely used in our country.

This method consists of the following. A large batch (100 g) of coal is placed in a special oven heated to 730° for a period of three and a half hours. Just like in an industrial coke-oven, the heating of the coal batch in a plastometric apparatus proceeds unilaterally, from its warm portion to the colder one, thereby forming a *plastic layer*, whose thickness varies with the brand of coal. The depth of this layer is commonly denoted by $y$ and determines the coking properties of coal and coal mixtures.

At the same time there is yet another value, $x$, which is obtained in the plastometric apparatus and which denotes *shrinkage* (volumetric contraction) of coal in the process of coking.

Experience in our principal coal fields has made it possible to determine the average plastometric indices of coal brands which are actually used more than any others in coking industry (Tables 7 and 8).

*Table 7*

**Plastometric Indices of Donets Coal**

| Plastometric indices | Г Gas | ПЖ Steam-fat, contract | ПЖ Steam-fat, strong | ПЖ Steam-fat, weak | К Coking | ПС Steam-coking, baking |
|---|---|---|---|---|---|---|
| Thickness of plastic layer $y$ in mm | 10-14 | 15-25 | over 27 | 22-27 | 16-22 | 5-15 |
| Shrinkage $x$ in mm | 25-40 | 15-35 | not over 15 | less than 12 | not over 20 | — |

*Table 8*

**Plastometric Indices of Coals from Eastern Regions**

| Plastometric indices | Gas of low fusibility | Gas | Fat | Coking fat | Coking | Coking meagre | Adjuvant fat | Adjuvant meagre | Coking, Karaganda |
|---|---|---|---|---|---|---|---|---|---|
| Thickness of plastic layer $y$ in mm | 9-12 | 13 and over | 25 and over | 13 and over | 13 and over | 7-12 | 7-12 | 5 and over | 11 and over |
| Shrinkage $x$ in mm | — | — | not over 16 | not over 28 | less than 28 | less than 28 | — | — | — |

3. *Investigations into petrographic composition of coal* is a very valuable means for forming an adequate judgement regarding its nature and technological properties.

There are four basic petrographic varieties in the most widely occurring banded coals, viz: *fibrous* (fusain), *dull* (durain), *lustrous* (vitrain) and *semibright* (clarain).

Outwardly, *fibrous* coal resembles charred wood. It is friable, crumbles to dust, has a black streak and soils fingers.

*Dull* coal consists chiefly of agglomerations of carbonised plant remains and is distinguished by its hardness and viscosity. When subjected to run-of-mine sizing, it is generally classified as large-sized.

*Lustrous* coal, by nature a solidified gelatinous mass, is distinguished for its conchoidal, vitreous fracture. It is rather friable and jointed and is generally sorted into undersize classes.

*Semibright* coal is less friable than lustrous and of greater viscosity.

In the petrographic study of coal microscopy plays an important role.

4. Widespread construction of concentration plants in the U.S.S.R. has brought in its wake considerable progress in preliminary investigations of coals for their amenability to *concentration.*

The accepted procedure is first to choose bed and production samples which are to be processed in accordance with State Standards. With the aid of screen analysis the sample is divided into classes by size, starting with lumps measuring over 150 mm, then 150-100, 100-50, 50-25, 25-13, 13-6, 6-3, 3-1, 1-0.5 and less than 0.5 mm.

Classes of 25 mm up undergo sorting with the picking of waste and pyrite aggregates (if they are there).

Classes of 100 mm and below, barring dust of 0.5-0 mm, are subjected to investigation in heavy liquids of diverse specific gravity, from 1.3 to 1.8. Classes of 6 mm up are usually scrutinised in water solutions of zinc chloride. Classes below 6 mm are examined in toluene (or benzene) carbon tetrachloride solutions.

The screening analysis of the production sample, the data obtained in large-sized class sorting and the results of the tests of medium and small-sized classes in heavy liquids determine the *granulometric* pattern of coal and ash content (if necessary, also that of sulphur) in individual size-grades and fractions of different specific weight. This information is used in selecting an adequate method of concentrating coal and estimating the size, character and power of mill equipment and the balance of concentration products.

5. *Dry distillation of coal* or its heating without air is done at different temperatures, depending upon technological aims. To obtain *metallurgical coke*, it is necessary to have high temperatures (1,000° and more). In this case the coke obtained is the basic ultimate product, while gaseous and vapour substances are considered *by-products*, although they are actually of extreme importance for the chemical industry.

Low-temperature coking is principally used for obtaining condensation products (primary pitches, etc.) and their subsequent processing into synthetic *liquid* fuels (benzene, ligroine, etc.). The *semi-coke*, which is the final product in this instance, is but of a secondary importance, though by its weight it predominates in the process. Here of major significance are the yield and quality of primary pitch. Semicoking proceeds at a temperature not exceeding 550° and, unlike high-temperature coking, the size of coal processed is of particularly great importance from the standpoint of the gas permeability of the stock in the semicoke oven.

In mining coal deposits, due account should be taken of the quality of coal to be extracted.

Of particular importance for the national economy is the coal utilised for the production of metallurgical coke, that is, that capable of coking independently or in mixtures with coal of other brands.

New deep mines in the Donets basin are sunk chiefly to work beds containing coals suitable for coking.

If a mine has beds or seams with coal of varying grades, the order and sequence of mining should be so planned as not to delay the extraction of coking coal beds by the exploitation of other seams. On the other hand, it would be wrong to leave unmined beds in already developed levels for lengthy periods of time just because their coal is not suitable for coking.

In mining coal measures, it is also necessary to consider the qualitative composition of coal when it is extracted separately from different seams. If this be the case, the order of mining should be arranged and the choice of suitable transport facilities made well in advance.

Spontaneous combustion of coal (Section 13) has also much to do with the qualitative pattern of coal. Anthracite, for example, is not self-igniting and its mining, therefore, does not require measures against underground fires, a thing that is mandatory when working beds containing spontaneously igniting coal.

Some coal (coking coals, coal used for power plants and burnt in powdered state) is ground prior to being used for industrial purposes and, therefore, can at least partially be obtained as fines, while other types of coal—for instance, gaseous, used in gas producers—should be drawn in large-sized lumps.

## 6. Hardness of Coal

Hardness of coal, that is, its resistance to mechanical agents, is of paramount importance for undercutting by coal-cutters or undercutting and slotting by combines, for work with mechanical picks, blasting, etc. The degree of hardness also determines the capacity of coal to break into pieces of different sizes and either to remain lumpy or disintegrate during trasportation and storage.

The methods of mining and mechanisation of stoping operations to be applied, with due account of coal hardness, depend on whether coal is to be obtained in large-sized lumps or is allowed to disintegrate.

The degree of coal hardness is rated in mining practice by including coal in a certain class (or *category*) of hardness. In the Donets basin, for example, it is as follows:

I. Hard anthracite (interfluent) without traces of cleavage and strong coal with no cleavage and jointings but with a high proportion of pyrite inclusions.

II. Anthracite with slight cleavage and coal of medium hardness, with no cleavage, jointings and inclusions.

III. Eutomous anthracite and soft coal with indistinct cleavage.

IV. Very soft eutomous coal.

The classes of coal hardness are established to conform to certain mining operations, such as excavation by combines, undercutting by cutters, boring by electric drills, etc. Thus, Class I includes coal for which the average feed rate of the Donbas and Gornyak combines is 0.27 metre per minute, and Class IV coal for which this rate is 0.7 metre per minute.

The hardness of coal is one of the major factors influencing production rates in different coal-seam mining operations.

It may also have a bearing on the selection of a mining method. For instance, in mining high seams with soft coal, the methods of exposing coal surfaces overhanging the working face are inadmissible. In coordinating the length of faces with the efficiency of available equipment, one should reckon with the hardness of coal since efficiency depends on it.

## 7. Cleavage of Coal

Cleavage is a property of rocks, particularly useful minerals, by virtue of which they detach or break more readily from the solid mass in one or several directions or planes than in all others. Cleavage is closely related to the jointing of rocks caused by intense tectonic processes. Donets miners have aptly dubbed cleavage "streams".

The cleavage phenomenon is related to the origin and geological history of rocks and its planes and their orientation are therefore governed by regularities incident to the geological structure of a given district. The direction of cleavage is usually uniform over large areas, especially if the strike and dip of the bed are constant. The position of cleavage planes in space may be characterised by the relation between azimuth and meridian and the angle of pitch to the horizontal plane. But it is usually only the angle between cleavage orientation and the strike of the bed that comes under consideration. Hence the expressions *cleavage with the bedding*, *cleavage with the dip* and *trans-*

*verse cleavage.* A clear-cut cleavage is usually called *distinct* or *obvious*, or, conversely, *blind joint.*

When coal was broken by hand, the relative position of the face breast and cleavage greatly influenced the labour productivity of the hand-cutter—cleavage parallel to the breast of the face facilitated breaking coal, while perpendicular cleavage complicated this operation. At present, when coal extraction is mechanised, the relative position of cleavage has lost most of its significance.

In certain cases, however, the position of cleavage planes deserves serious consideration on account of safety. When high-pitching and, particularly, thick beds are mined, large coal blocks are apt to become detached along the cleavage planes. This hazard can be eliminated by the proper sequence of coal extraction and adequate timbering at production faces.

## 8. Size of Coal

In many instances, the size of coal is an important factor of industrial utilisation.

As stated above, coal for gas producers should be in lumps of definite size. Fine lignite is unsuitable for locomotive fireboxes or stationary grate furnaces. The same applies to long-flame coal, whose particles do not fuse in burning.

Conversely, the size of lumps is of no importance for coal used in the production of metallurgical coke, since it has to be ground prior to coking anyway. For coal burnt chiefly in the boiler furnaces of power stations in powdered form the size of raw coal delivered by the mine is of no consequence either.

Coal whose size is essential for industrial utilisation, and which consequently has to go through sizing and screening, is classed as follows (Table 9).

*Table 9*

**Grades of Brown, Long-Flame, Gas and Anthracite Coal**

| Grades of coal | Size in mm | Designation of coal grades | | | |
|---|---|---|---|---|---|
| | | Brown | Long-flame | Gas | Anthracite |
| Slabs | +100 | БП | ДП | ГП | АП |
| Run-of-mine | —100 | БР | ДР | ГР | АР |
| Large-sized nuts | 100-25 | БК | ДК | ГК | АК |
| Small-sized nuts | 25-13 | БМ | ДМ | ГМ | АМ |
| Flaxseed | 13-6 | БС | ДС | ГС | АС |
| Flaxseed with coal dust | 13-0 | БСШ | ДСШ | ГСШ | АСШ |
| Coal dust | 6-0 | БШ | ДШ | ГШ | АШ |

The "large-sized nut" grade sometimes includes a special grade of gas-generating fuel, its size ranging from 50 to 25 mm (АГ, etc.).

From 25 mm down the value of coal grades diminishes progressively. Besides, State Standards provide for rebates in prices of coal whose grades contain fines in excess of established proportions. Determination of fines in graded fuel is effected by *screening* samples through a set sieve. This operation is called *sizing assay* or *screen test*.

The size of coal broken in the mine depends on its hardness, jointing and cleavage, as well as mining methods and haulage. Coal whose size is important for utilisation should be mined and transported by methods minimising their disintegration. One essential disadvantage common to mine combines is that they yield a large proportion of *coal dust*. The designers, therefore, should concentrate their efforts on building machines operating not on the principle of cutting, like ordinary coal-cutting machines and mine combines, but on that of *broad shearing*, without reducing broken coal to dust.

Lignites, when stockpiled in the open air, are liable to disintegrate and crumble. Therefore, they should be kept in storage for as little as possible and stocked in low piles, this also being of importance for preventing their spontaneous combustion.

## 9. Properties of Wall Rocks

Country rocks enclosing coal seams are made of silt, sand, rubble, etc., which lay under and over the agglomerations of vegetable matter during the formation of deposits. In the course of the lengthy period of their geological history, these mineral sediments have been subject to changes (*diagenesis*) which have, generally, led to the formation of rocks. The rocks specific of most coal fields dating from the carbonaceous age are *sandstone*, *clay* and *sandy shales* and *limestone*.

Of these sandstone and limestone are the strongest.

Since silt and sand, as well as other mineral particles, sometimes mixed during the accumulation of initial mineral matter, there are rocks which are a cross-section between the above-cited typical rocks of coal deposits. Shales, found among the latter, which contain a high proportion of carbonaceous substance, are called *coaly shales* or *ampelites*.

The extent of loose rock "lithification" was not dependent only upon the duration of its geological life, but also on the effect of tectonic factors. The Moscow coal fields, for example, are of approximately the same geological age as the Donets basin, but reposing as they do upon the Russian continental plateau, they have not undergone orogenesis while the changes in the structure of country rocks enclosing these coal deposits have been relatively slight. Therefore, the coal measures of the Moscow basin are overlaid with *clay* and *sand*, which have not been transformed into clay shales and sandstone.

Friability is also a feature specific of rocks enclosing the brown coal deposits of the Tertiary Period (Ukraine, Bashkiria).

It should be noted that, on the suggestion of the outstanding geologist, Academician M. Usov, rocks analogous to clay shale in the Donets basin are quite often called *argillites*, and sand shale—*aleurolites*. However, for practical purposes, this is of no significance.

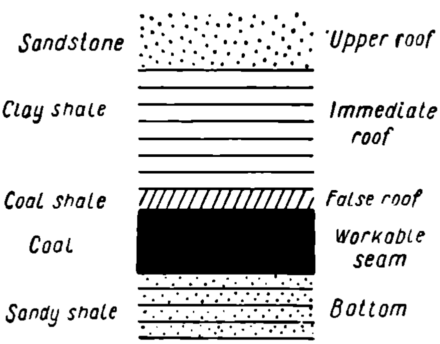



*Fig. 123.* Diagrammatic representation of wall rocks in a coal seam

Everything said about cleavage in coal beds also applies to the rocks enclosing them.

The properties of wall rocks play a major role in the selection of the methods of mining and stoping.

The degree of the rigidity and jointing of wall rocks, the direction of cleavage and fissures, the capacity of rocks to become detached and fall in lumps or large blocks, the tendency suddenly to cave in or sag gradually—all this is often of decisive importance for choosing an adequate method of mining and stoping, a fact that will be repeatedly emphasised in the chapters below.

In gently dipping deposits the most significant factor in mining is the properties of the roof, though the floor's features must sometimes be considered too; for instance, when it is apt to *heave*, that is, to rise in production faces and in the workings in general. *Heaving* and *bulging* of the floor rocks in mine workings is partly explained by the pressure exercised by the surrounding rocks and partly by the absorption of moisture by argillaceous rocks and their concomitant swelling. In the case of a high dip, the properties of the floor, which is apt to slide and cave in, are in a large measure responsible for some of the peculiarities of mining methods adopted.

When a thin layer of easily caving rock lies immediately over the mined bed it is called *false roof*. In coal seams, the false roof is frequently made up of carbonaceous or weak clay shale. The stronger rocks overlying the false roof are called *upper* or *main* roof. Often there are only *immediate* and *main* roofs. The first is a mass of rocks, usually several metres thick, which falls in the mined-out area the moment the roof caves in by itself or is made to do so. A typical example is given in Fig. 123. Here the main roof includes sandstone, the immediate one—argillaceous shale and the false roof—coaly shale. Quite often when there is no false roof at all, the immediate roof made up of stable solid rock may be regarded as the main roof. The layer of strongly coherent rock lying immediately over the coal bed is frequently called *slab* roof.

The capacity of rocks to adhere firmly (or loosely) to the roof of working faces causes them to be characterised by such terms (rather vague) as *stable* or *coherent*, *strong*, *medium stable*, *unstable* and *weak*.

A *short* roof is one of sand or weak shale crumbling into very small particles. With a short roof, to avoid contamination of coal, it is necessary not only thoroughly to "tighten" the roof during timbering, but sometimes even to leave a *ceiling* or a *coal layer* in the roof about 10 or more cm thick.

We shall return to the properties and behaviour of rocks in their capacity of roofs in working faces in Chapter XI.

## 10. Spatial Relationship of Seams in the Series

In the mining of coal series, displacement of rocks following the extraction of the mineral from one of the seams may affect mining conditions in the neighbouring ones. On account of that, *contiguous* beds should be worked in a definite sequence so as to prevent them from *undermining* each other.

This complex problem is very largely related to the peculiarities of the mining methods now in use. Therefore, it is advisable to discuss it in greater detail in Chapter XXII.

## 11. Mining Depth

As the depth of mines increases, one may expect certain changes in the nature of rock pressure, in rock temperature, and in their water and gas-bearing capacity.

There is no doubt that in the earth's crust rocks are in a state of *stress* and that in a solid block of rocks these stresses increase along with the depth. As mine workings are advanced, however, there occurs a redistribution of the stresses formerly existing around them. Therefore, the new stresses, whose distribution depends on initial tensions, the shape of mined-out spaces and the physico-mechanical properties of rocks, are no longer proportional to the depth and, as a matter of fact, their distribution is extremely complex.

This phenomenon is, apparently, related to the important fact that in conditions of rock pressure prevailing in development workings and stopes at a depth of about 1,000 metres, now reached in some coal mines, there has as yet been no distinct manifestation of it observed that could be ascribed to the increased depth of mining. On the other hand, it can hardly be denied that below 1,000 metres there may arise pressures which are not encountered at lesser depth from the surface.

In the earth's crust the *temperature of rocks* increases along with depth. It rises by 1° approximately every 30-35 metres, sometimes considerably more (geothermal gradient is the rate at which the temperature increases per 1 km of depth). But these are but average figures, since accurate recordings of temperatures in deep boreholes show that in the southwestern part of the Donets basin, for instance, the geothermal gradient ranges between 27.3 and 70.4 metres. In the same area it has been ascertained that the mean temperature of rocks from the ground surface is as follows: at the depth of 20 metres—15.1°; 700 metres—29.8°; 1,200 metres—44.1°. The temperature taken at the depth of 1,450 metres in a borehole of the Donets basin in 1952 was 54°, which corresponds to a geothermal gradient of 31.5 metres. The rock temperature increased in direct proportion to the depth. Hence, to find the approximate temperature of rocks, divide the depth of mining by the value of the geothermal gradient and add the mean temperature prevailing in the given area over several years (for example, about +8° in the Donets basin).

The temperature of the air in *mine workings* is lower than that of the rocks on account of the cooling effect of ventilating currents.

High temperature in stopes makes miner's work painful and unhealthy, and the result is a drop in efficiency. Therefore, special plants are installed to cool the air *entering* underground workings.

The relation between the gas- and water-bearing capacities of rocks and the mining depth will be discussed below.

## 12. Gas-Bearing Capacity of a Deposit

Each mining method should be elaborated in such a way as to ensure adequate supply of pure air to the stopes of development and production workings.

Of the gases which mix with the atmospheric air of normal composition, the most important are detonating gas, carbon dioxide, carbon monoxide and gases released by blasts.

Sometimes the atmosphere in coal mines also contains hydrogen sulphide.

*Detonating gas*, or *firedamp*, is methane, sometimes with slight admixtures of other hydrocarbons and hydrogen.

There are three basic ways in which detonating gas escapes into the mine atmosphere, viz.: 1) placid outflow from coal or wall rocks, without any outward signs; 2) outflow in the form of gas jets from cracks and fissures with audible sound effects (*fumaroles*); 3) *sudden* *outrushes* or *instantaneous outbursts of coal and gas*, that is, gas discharges accompanied by ejection of finely broken coal.

Mines, in which firedamp is formed, are called *gassy* or *fiery*. The mines are classified in accordance with the abundance of gas they contain, this depending on the volume of gas evolved (see Table 10).

*Table 10*

**Categories of Mines According to Abundance of Gas**

| | I | II | III | Super-category |
|---|---|---|---|---|
| Volume of methane formed per ton of average daily output (relative gas abundance) in cu m | Up to 5 | From 5 to 10 | From 10 to 15 | Over 15, or mines working seams made hazardous by outbursts of coal and gas and by fumaroles |
| Minimum volume of air per ton of average daily output, in $m^3$/min | 1 | 1.25 | 1.5 | See note |

*Note:* For super-category mines the proportion of methane in the total return current should not exceed 0.75 per cent, nor be less than 1.5 $m^3$/min per ton of average daily output.

*The relative abundance of methane in mines*, that is, the volume of this gas in cubic metres forming per ton of coal extracted (daily average) tends to increase with depth. Investigations carried out by G. Lidin show that in the Donets coal fields the ratio is as follows:

| Depth of mine, in metres | Relative abundance of methane per ton of coal output, in cu m |
|---|---|
| Up to 150 | 1.2 |
| 150-250 | 5.7 |
| 250-350 | 9.5 |
| 350-450 | 11.3 |
| 450-550 | 16.3 |
| 550-800 | 20.0 |

The figures above are averages for the basin as a whole; the proportion of methane in some mines deviates considerably from these averages.

According to Academician A. Skochinsky, the discharge of methane in the future mines of the Donets basin at depths of 1,000-1,500 metres may be expected to reach 50-80 cu m per ton of daily coal output.

G. Lidin estimates the following average distribution of methane evolved in the mines of the Donets basin: from working seams—60 per cent, contiguous beds—10 per cent, from enclosing country rocks—30 per cent.

In gassy mines the adopted mining methods should envisage as few dead faces as possible, especially in ascending workings. Being almost twice as light as atmospheric air, evolving methane tends to occupy the upper portion of an ascending working, in other words, it accumulates right at its face. The study of causes leading to explosions in coal mines reveals that in most cases gas ignites at the face of an ascending working, where it amasses on account of inadequate ventilation. From such a face the explosion may spread through coal dust all over the mine and cause a major catastrophe.

Generally speaking, the presence of firedamp requires active and intense aeration of mine workings. Whenever possible, individual sections of a mine field should be ventilated by separate air currents to prevent the consequences of a possible explosion or shortcomings of ventilation within one section from adversely affecting the others.

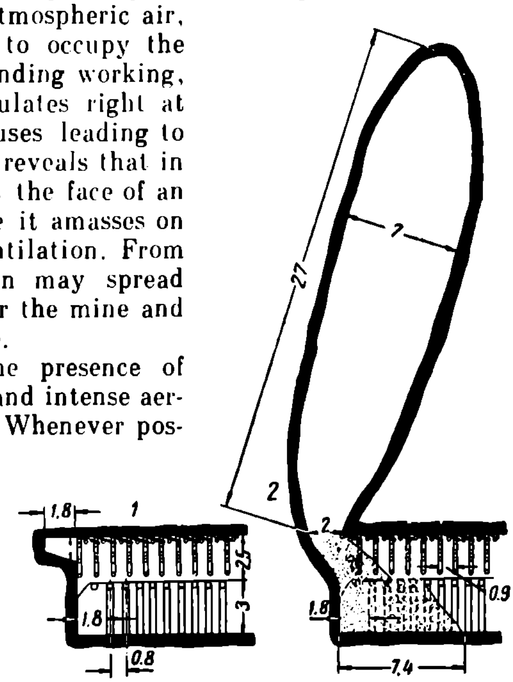


*Fig. 124.* Shape of a hollow space formed after a sudden outburst of coal and gas

Insufficiently aerated hollow spaces in mined-out and abandoned areas of gassy mines present a definite hazard. They may become a scene of firedamp accumulation and if barometric pressure drops abruptly or the roof caves in over an extensive area this gas can penetrate into active workings.

One of the most dangerous forms of methane evolution is *sudden outburst of coal and gas*. The amount of coal ejected in these instances ranges from a few to hundreds and even thousands of tons, this being accompanied by discharges of large and sometimes huge volumes of gas. A sudden outburst produces sound effects of diverse intensity. After such an outburst a hollow space or cavity of a peculiar shape forms within the coal seam (Fig. 124). The drawing is illustrative of the case of instantaneous outrush which occurred at the face of a strike entry running along the Mazur seam in the Krasny Oktyabr Mine (Donets basin) on February 15, 1951. The position of the face prior to the outburst is shown in Fig. 124, *1*. When he heard the cracking of breaking coal, followed by a strong shock in the coal block and sounds resembling machine-gun fire, the miner working at the face left it. Then came an outburst of coal and gas. The amount of coal ejected totalled 71 tons. The force of the outburst was so great that it knocked out a portion of the face timbering and threw the coal 7.4 metres away from the face. The hollow space formed in the seam is depicted in Fig. 124, *2*.

According to the opinion voiced by Academician Skochinsky, the sudden outburst of coal and gas is a consequence of an avalanche-like progressive dislodgement of coal under the effect of rock pressure and gas contained in coal attended by a dynamic effect involving ejection of coal and its disintegration, unusually intense discharge of gas within a short time and formation of characteristic hollow space in the seam.

Sudden outbursts should not be confused with the following three phenomena:

sudden downfalls or *inrush* of coal, accompanied by gas discharges (sometimes very abundant), but without too high a head;

crushing and *squeezing* of coal, with comparatively slight evolution of gas and with the coal being thrown a short distance;

instantaneous *gas rushes* without coal outbursts but with splits in the roof and the bottom and the formation of one or several fractures and cracks (sometimes quite extensive).

The causes and mechanisms of sudden outbursts have not been studied thoroughly as yet. The main factors underlying a sudden outburst are apparently the following.

When a mine opening is driven in a coal seam saturated with gas under pressure and containing soft or, at any rate, not very strong coal, rock pressure causes the coal at the face to split and disintegrate; it is then detached and thrown off by the pressure of escaping gas. In steeply dipping seams, the downthrow of coal is facilitated by its own weight. This process gains momentum extremely rapidly—*avalanchelike*—from the face into the coal massif until the pressure of gas subsides following its escape and the cavity of the outburst acquires a stable pearlike shape. Sudden outbursts, with rare exceptions, originate at the depth of 200-300 metres from the surface. This is due to a certain amount of *degassing* in seams caused by the leakage of gas to the ground surface in the past. It has been observed that in high-pitching seams in the Donets coal fields instantaneous outbursts are very rare in the upper third of the level intervals. The reason is degassing of coal and wall rocks, with the gas escaping into an upper entry, which first serves as a haulage working and then as an airway.

Sudden outrushes usually (but by far not always) occur at geologically disturbed sites, where coal, it may be presumed, was subjected to high stresses during tectonic processes and subsequently to crushing and breaking.

In Soviet coal mines sudden outbursts occur in the Donets coal fields, especially at the deep levels of mines working high-pitching coal measures in the area of the Main Anticline, in the Urals (Yegorshin anthracite district), in the Kuznetsk coal fields (Severnaya and Tsentralnaya mines near Kemerovo) and in the mines of Suchan (Far East).

In mines with seams susceptible to sudden outbursts of gas and coal, in addition to the safety measures generally taken in mines against methane hazards, the special precautions given below are obligatory, their purpose being to prevent sudden gas and coal outrushes and ignition of released gas, facilitate the rescue of men and eliminate the damage:

1) adoption of mining methods requiring minimum development work, with maximally regular shape of production faces devoid of pointed projections;

2) working of *protective* seams; in mining a series of seams, some of which are subject to sudden outbursts, the first to be extracted are the contiguous seams overlying and underlying them and "protecting" the hazardous ones;

3) breaking of coal in development and production stopes solely with the aid of blasting and rotary drilling, without the use of percussion instruments and machines;

4) blasting of drill-holes in-between shifts, when there are no workers in the mine, effected from the surface or special shelter-rooms;

5) the use of so-called *concussion* blasting, with the number of holes drilled in excess of that actually needed for the breaking of coal and with larger charges than usual;

6) measures designed to facilitate the rescue of men caught in a mine by a sudden gas outrush, viz.: separate ventilation pattern; arrangement of shelter-rooms; distribution of self-rescuers among workers; placing of miner's electric lamps-guides along the routes of men's possible escape, etc.;

7) drilling of advance exploration holes (up to 10 metres deep) in the faces of development workings.

The volume of methane liberated in coal mines may reach proportions that necessitate its *catchment and practical utilisation*.

Organisation of catchment is easiest in the case of fumaroles active in a working that can be isolated from the rest of the mine. The site with fumaroles is isolated by a seal, from behind which gas is brought by a pipeline to the surface.

Yet another method is *drainage* and *aspiration* of the gas from coal beds. A pilot gas-aspirating plant was commissioned at the Severnaya Mine in the Kuznetsk basin in 1951. A series of drainage boreholes up to 50-60 metres long were drilled along the Volkovsky seam, which is very rich in gas. To bring methane up, boreholes were connected hermetically with a pipeline which was linked on the surface with a vacuum pump. An average of 1,000 cu m of methane was aspired every 24 hours. At present degassing plants operate in a number of mines in the Kuznetsk and Donets basins. In some instances the gas thus caught is utilised as a high-grade fuel.

The importance of gas draining and aspirating plants lies not only in the fact that they open up new possibilities for utilising methane, but also in the fact that catchment of gas reduces its volume and pressure in coal seams (this is particularly important in working seams susceptible to sudden outbursts) and decreases amounts escaping into active mine workings.

In some foreign countries sudden outrushes occur where the gas is not methane but *carbon dioxide.* No such sudden outbursts have been registered so far in the U.S.S.R. Nevertheless, we should reckon with possible carbon dioxide accumulations in our mines, which can largely be due to decay of timber in mined-out areas.The carbon dioxide level in the mine atmosphere should not exceed 0.5 per cent, the only exception being made for the workings with a common return current, where it is allowed to be 1 per cent.

The appearance of *carbon monoxide* and *fire gases* in the atmosphere of mine workings is a consequence of underground fires.

## 13. Spontaneous Combustion of Coal

The main reason for spontaneous combustion is *oxidation* of coal with atmospheric oxygen. Secondary factors contributing to spontaneous combustion are the presence of pyrites in coal and, to a certain extent, moisture content. The intensity of oxidation is directly related to the physical and chemical nature of the coal, the degree of its breakage and the rate of air inflow to its accumulations. If coal is in a solid block, its contact surface exposed to the atmosphere is small and there is practically no oxidation. If, on the other hand, coal is split by fissures or broken into smaller pieces, its surface exposed to contact with atmosphere increases immensely and this may cause faster oxidation. In the latter case the temperature goes up and this, in turn, contributes to the intensity of coal oxidation. The increase in temperature may attain a degree where coal becomes incandescent and burns with an open flame. The greater the natural softness conducive to the breaking up of coal the greater the hazard of its spontaneous ignition. Also dangerous in this respect are areas with geological dislocations containing crushed coal.

The main prerequisite for spontaneous combustion is, therefore, the loose state of coal. Consequently, particular hazards are presented by crushed pillars and amassments of coal fines in worked-out areas. Since coal losses in the mining of high seams without filling are greater than those in the case of thin ones, the fire hazard in the first instance is much bigger.

Self-ignition is a property more or less common to almost all kinds of mineral coal. Although in practice coal seams are subdivided into self-igniting and nonself-igniting, such classification is highly conventional. The absence of spontaneous combustion in conditions where coal losses are kept down and there are no crushed pillars by no means warrants considering a seam nonself-combustible.

Combustibility applies not only to coal but to gangue containing carbonaceous matter too.

There is almost no danger of self-ignition of coal with a minimum level of volatile matter, such as anthracite. But in the case of lean coal, spontaneous combustion may be quite intensive.

Experience, at any rate, has shown that high seams in most coal deposits in the U.S.S.R. are subject to spontaneous combustion. One exception is the Yegorshin anthracite deposit (Urals). In spite of the faulty nature of the occurrence, the softness of anthracite and high exploitation losses of coal, no fires due to spontaneous combustion have so far been recorded there, although these mines have been worked for several decades now.

No universal measures have so far been devised to combat underground fires caused by spontaneous combustion. *Active fire-fighting* is resorted to only when the area affected is small and accessible. Fires in inaccessible areas are put out by the *isolation* method. To do this, the suspected fire-stricken section is sealed off by airtight *bulkheads*, and the fire gradually dies down because of lack of oxygen. This method is adequate for thin and medium thick seams, but in the case of high seams the ultimate extinguishment of a sealed-off fire requires *silting*, that is, filling the fire-stricken section with a liquid clay solution through boreholes. It is also necessary constantly to watch temperature and prevent outside air from penetrating into the fire-stricken area from the surface through crevices and holes. The latter should be regularly filled and tamped with clay. Experience, unfortunately, shows that these measures are not always effective and adequate. When attempts are made to unseal the isolated fire-stricken section it is often found that the fire has merely been subdued but not extinguished and the inflow of fresh air again sets it ablaze. Fire-fighting measures are thus a costly item of expenditure, hamper normal activity in the mines, divert the attention of the supervising technical staff from other production problems, require a considerable labour force and yet do not always bring the desired results. Large coal reserves, developed for extraction, become irretrievably lost in fire-stricken areas.

Underground fires are especially dangerous in high-dipping deposits. Let us assume that a fire has broken out in one of the levels which is then sealed off by bulkheads. The working of the underlying level may "undermine" the fire-stricken section, that is, cause the pillars surrounding it and even the bulkheads to develop fissures enabling air to circulate. The influx of oxygen will not only set the smouldering fire ablaze again but also cause it to spread to the workings of the underlying level. Underground fires are known to have a tendency to spread in a direction opposite to the movement of fresh air currents. With the appearance of the above-mentioned fissures and the increase of temperature, the air currents in the fire-stricken area will, generally speaking, move upward.

Men engaged in putting out and sealing off underground fires very often have to wear respirators. The access to the site of fire for fire-fighting teams in such masks is facilitated by the numerous break-throughs usually available in the mining of the first levels of high-pitching or sloping deposits. But with stoping proceeding at deeper levels the number of openings communicating directly with the surface becomes progressively smaller, and this greatly complicates fighting underground fires in deeper mines.

Hence, the hazard of spontaneous coal combustion should be regarded as a major factor in selecting mining methods. Self-igniting seams have to be worked with minimum losses, and in thick high-dipping beds this can be achieved only by a complete fill. This, in turn, exerts a decisive influence on the nature of mining itself. Self-igniting seams should be worked with maximum speed and by sections, which can be isolated rapidly from each other, a fact which also distinguishes the mining method to be chosen.

## 14. Water-Bearing Capacity of a Deposit

The water-bearing capacity of a deposit is a factor which necessitates elaboration of certain specific features for the method of mining it.

The presence of water in workings reduces labour efficiency and is, moreover, dangerous. Men engaged in wet stopes easily catch cold; when the floor and timber-pieces are wet and slippery, miners are apt to fall and hurt themselves; by absorbing water, clay wall rocks become weaker and more liable to caving. In wet stopes the law provides for, or allows establishing, in certain conditions, a reduced workday and, depending on the amount and nature of water inflow, lower production rates.

For this reason measures should be taken to divert mine water from active faces.

When the bottom is rough and the occurrence of a bed is nearly level, its extraction in highly acquiferous deposits should be arranged so as to make the drain ditch now run above coal in the sink holes of the bed, now become rather deep.

Acquiferous friable rocks require deep *drain* ditches capable of holding large volumes of water.

In mining highly acquiferous deposits, particularly those in which sand and particles of other rocks are entrained by water, one should take special steps in production faces in the form of definite measures for roof control and the setting up of bulkheads. But it is still better *systematically to dewater* such deposits prior to starting stoping operations in the section. To protect underground workings from inrushes of water from the underground or surface reservoirs—which, for some reason, can not be drained—safety pillars are to be left under these reservoirs. There have been instances of deposits being mined even under the sea bottom.

Below we shall meet with the description of technical methods used in mining acquiferous deposits.

## 15. The Effect of Mechanisation on the Selection of Mining Methods

Mechanisation of coal production is a factor not to be underestimated in the selection of mining methods.

Cutting machines, mine combines, mechanical picks, conveyers, transport facilities, etc., have already wrought immense changes in the mining methods formerly used in the same natural conditions. Ventilating equipment, good lighting, adequate means of communication (telephone, signalling systems) have also completely transformed working conditions in the faces. Thanks to mechanisation, and wide application of mechanical energy in general, there are now faces 200 metres and more in length. The use of cutters and combines have made it possible to have linear faces in steeply dipping seams. The list of such examples could be continued.

Therefore, in planning mechanised coal production, one should very carefully select a mining method best adaptable to the available equipment and, conversely, to choose the machinery most suitable to the conditions from among that actually in use. In this respect, *mining methods and systems of mechanisation are closely interrelated.*

Another basic factor to be considered in selecting a mechanised method of mining is *complex mechanisation*, which coordinates the operations of all mining machines in a given section. The sequence of operations by machines linked with each other technically or organisationally (for example, cutter or combine, push conveyer in the face, belt conveyer in the entry and the incline) should cause no "bottlenecks" in the work of machines or continuity of operation which might result in delays in the work of other equipment or hold up other operations.

There is no doubt that wider use and improvement of the existing types of machines for undercutting and cutting coal, its tramming in production faces and along haulageways and introduction of new types of equipment, whose employment is imminent, *will lead to the appearance of entirely new mining methods, in addition to those in existence now.*

The factors above should be taken into account in selecting mining methods. The choice of the method, generally speaking, is a difficult task which requires a great deal of caution and considerable experience for many of the factors cited are in opposition to each other. The main aim in selecting an adequate mining method in the conditions of the Soviet national economy is *achievement of safety and maximum labour efficiency along with the improvement of working conditions by the employment of machines and suitable mining methods, as well as reduction of mineral losses to the minimum.*

Below, in describing each individual mining method, we shall touch upon the influence exerted by all or some of the above-mentioned factors.

CHAPTER X

# GENERAL SURVEY OF COAL SEAM MINING METHODS

## 1. A Note on the Classification of Mining Methods

*Classification* of mining methods is based on a number of characteristics whose essence can be understood only after a study of their individual types. It is for this reason that the classification of mining methods itself is given in Chapter XXIV at the end of the book.

## 2. Fundamentals of Breast-Stoping Methods of Mining

We have learned above that in opening coal seams mine fields, as a rule, are subdivided into levels.

In the simplest case, the stoping of coal in each flank of a level with inclined height $h$ can be carried out in one *straight stope or face ab* (Fig. 125).

From this production face (wall) there should be two absolutely free exits, one leading to lower (haulage) entry *ac* and the other to upper (ventilating) entry *db*.

Coal extracted from the wall is delivered to the haulage entry at point *a*, whence it goes to the hoisting shaft.

The direction of the working face *advance* (at the rate of $L$ [m] per year) is shown in Fig. 125 by an arrow. A *mined-out space* remains behind the production face as a result of stoping. Haulage entry *ca* and ventilating entry *db* are protected from rock pressure by coal pillars or back filling (Fig. 125).

This method of mining is distinguished by the fact that there is one production stope or wall, whose length (if one ignores the height of stub pillars) is equal to the level interval. This method, therefore, is called *longwall mining*. In the author's opinion, it would be more convenient to regard this method as a modification of breast stopping (see below).

Fig. 125, *I* illustrates an instance of advance mining of a level. But it is also possible to work a longwall (just as it is in the case of any other mining methods described below) in the retreating order of extraction (Fig. 125, *II*).

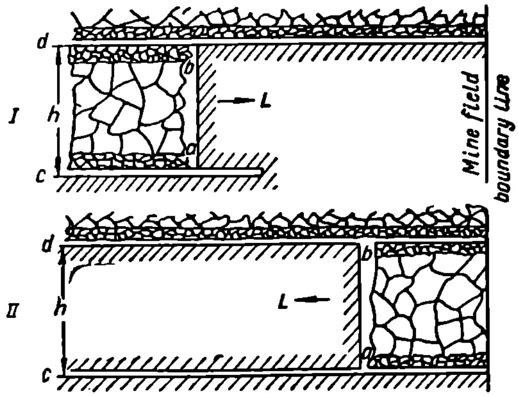



*Fig. 125.* Longwall mining methods
*I*—advance order; *II*—retreat order

Since in the latter case no development openings are made in the level delimited by haulage and ventilating entries, this presents sufficient ground, the author thinks, for including the retreat method of working a longwall in the group of breast-stoping methods too.

Mining a longwall is an extremely simple operation. But it is not always possible to work a level by a wall whose face extends over its entire height, and then stoping in the level is done by two (Fig. 126) or several walls. Communications between these walls and main haulage entry *ca* are maintained in the following manner.

*Incline gf* and *intermediate entry gb* are arranged *in the worked-out area.* Coal won in *upper* wall *a'b'* is passed along the face down to the intermediate entry, then hauled along to incline *gf*, whence it goes to haulage entry *ca*. Manway *eh* is usually driven over the entire height of the level, parallel to the incline. If the type of haulage adopt-

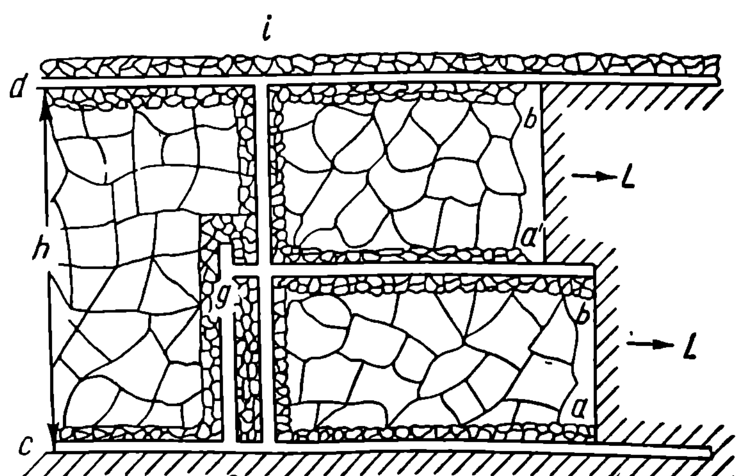



*Fig. 126.* Continuous breast stoping

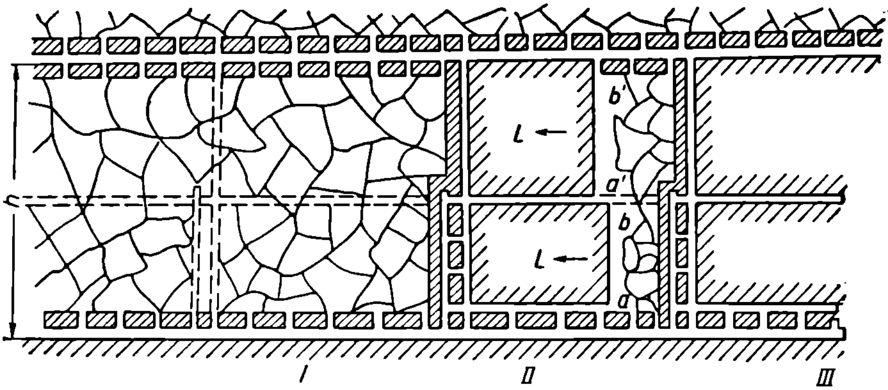



*Fig. 127.* Pillar method of mining

ed in the incline does not interfere with the passage of men, there may be no need for a special manway, but a manway leading to the ventilating entry must then be provided in the upper sublevel.

Like main strike entries, inclines, manways and intermediate entries, made and maintained in mined-out space, are protected by coal pillars or mine-fill.

As working faces move away from the incline, the intermediate entries grow progressively longer, and this increases expenditure for their upkeep and transport costs. Since intermediate entries pass through mined-out areas, their maintenance represents a substantial item of expenditure. To reduce it, new inclines are arranged in worked-out areas as the face advances, while the old ones are abandoned, as are the sections lying between them in the intermediate entries. In Fig. 127 abandoned mine workings are depicted by dash lines.

As Fig. 126 shows, there are no development workings in front of the *advancing production faces* in the level delineated by the main level entries. Mining methods possessing this distinctive feature are called *breast-stoping*. A typical example of such a method is given in Fig. 126. Inasmuch as working by longwall also conforms to that, it should be regarded as one of the variants of breast-stoping.

The portions of a level lying between adjacent entries (main and intermediate) are called *sublevels*.

Correspondingly, a wall whose length (as previously, with a possible corrective for the height of pillars) is equal to the height of the sublevel is sometimes called *wall-sublevel*.

Hence the method represented diagrammatically in Fig. 126 is distinguished by *division* into *sublevels*, while there is no such subdivision in that of longwalls.

The portion of a level lying between the neighbouring inclines is termed *production block*. When the working face within a level (in mining a longwall) or within a sublevel (if the level is divided into sublevels) forms an uninterrupted line it is said to be *continuous*.

There should be a clear-cut distinction between two concepts:

1) breast-stoping with no development workings driven ahead of the production face *front* in the level (this condition does not apply to the main strike entries); and

2) a continuous face whose definition has just been given.

Continuous production faces (longwalls) may thus be encountered in the pillar methods of mining too.

### 3. Definition of Pillar Methods of Mining

One major disadvantage of breast-stoping is the necessity of driving and maintaining mine workings in the midst of worked-out areas. Generally speaking, this shortcoming is felt the more strongly the thicker the coal bed. Because of this, *pillar* methods are used. A typical example is given in Fig. 127.

The main feature of the pillar methods of mining is that the development workings are driven and maintained not in the midst of mined-out areas, but in the mass of solid coal. To achieve this, all necessary development workings are prepared in the production block before stoping operations are started in it.

Fig. 127 shows that, while production block *III* is in the stage of *development*, block *II* is being stoped, and block *I* has already been mined out.

With the pillar method, a level can be divided not only into two sublevels (see Fig. 127) but into three and more. In this method, each production block represents a *pillar* or *panel*, that is, a solid mass of coal delimited for subsequent extraction by stoping. Pillars shown in Fig. 127 are rectangular, extending along the strike. Therefore, this is a method of long-pillar mining along the strike. There also are some other, now almost fully abandoned, methods of mining by *long pillars up the dip* and *pillar-and-stall* (Chapter XIII).

The above-cited modifications of breast-stoping (longwalls) and mining by long pillars along the strike (Fig. 127) are distinguished by continuous rectilinear production faces (walls) *ab*, *a'b'*. Such faces are of exceptional significance in the case of modern methods of mechanised stoping with the aid of mine combines or cutting machines (and also coal planers) and conveyers used in slightly inclined and sloping coal seams. Working faces may also be rectilinear in high-pitching seams, when extraction of coal is done by cutting machines, combines or planers. But, as will be seen below, the present practice still provides for a wide use of pneumatic picks and *overhand stopes*.

## 4. Definition of Combined Methods of Mining

The principal distinction of breast-stoping is the absence of development workings in front of working faces (the only exception being the main level entries), while a feature specific to the pillar or panel method is the preliminary cutting of coal massifs into pillars or panels, which are then stoped out.

There may be methods based on the following principle: stoping is started directly in production blocks, but at first the working faces are not extended over the entire coal area in the production block, but advanced so as to leave coal pillars between worked-out areas, which are later recovered.

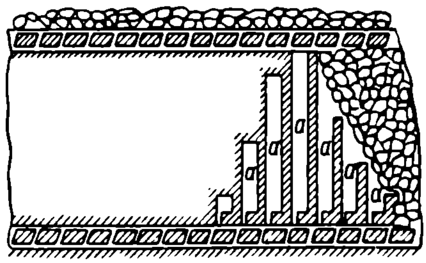



*Fig. 128.* Room-and-pillar method of mining

Thus, with these methods, extraction of coal in the first stage does not involve any development work. In this stage it is done by following the principle of breast-stoping, while the second stage—pillar recovery—is typical of the pillar methods of mining. These methods, therefore, are termed *combined.*

A typical example of this group of mining methods is that of *pillar-and-stall* mining, which has by now almost completely lost its significance (its brief description is given in Chapter XIV).

The combined group also includes the *room-and-pillar* method (Fig. 128). Its principal feature is that at first stoping operations proceed at the room faces (*a*) of a production block, usually about 4-7 metres wide. *Rib pillars* of coal are left in-between and are then recovered in a *retreating order*, that is, in the general direction opposite to that in which stoping was carried on in the rooms. As will be seen later, the rooms are made insignificantly wide with the view to reducing roof pressure. Because of that, the room-and-pillar method requires *narrow* or *short* production faces, as distinct from the other methods cited above, which require *wide* or *long* faces.

If rib pillars were not recovered after the rooms had been worked, the method would be called *room-mining*. However, the method of room-mining is no longer employed in working coal beds.

## 5. A Note on Methods Used in Mining of High Seams

The methods employed in working thick seams are constructively more complex.

One of the main principles underlying the mining of thick coal beds implies their division into *slices* usually about 2.5-3 metres thick, that is, seemingly into individual seams of medium thickness. Each slice is mined by methods which are used mostly in working seams of medium thickness, particularly with continuous rectilinear faces in slightly inclined deposits, and sometimes with stepped ones in steeply pitching seams. In this respect, drawing of coal from continuous faces is also of great importance in mining high seams by the slicing method.

Thick beds are also mined by methods envisaging no division of seams into separate slices.

## 6. Sequence of Describing Methods of Coal Mining

Henceforth we shall keep to the following order in giving detailed description of coal-mining methods.

Since it is in the continuous faces that stoping operations by the breasting (longwall) and pillar methods of mining thin and medium-thick beds, as well as by the methods of working high seams by slicing, are chiefly practised, these operations will be covered by a special chapter (XI).

The group of breast-stoping methods will be discussed in Chapter XII, pillar methods in Chapter XIII and combined methods in Chapter XIV. Methods used in mining very thick coal beds are dealt with in Chapter XV.

Issues concerning the working of groups of two or several contiguous seams have to be discussed all together, and this is done in Chapter XXII.

CHAPTER XI

# STOPING IN A CONTINUOUS FACE

## A. SLIGHTLY INCLINED AND SLOPING SEAMS

### 1. Rock Pressure in a Continuous Face

The *active stope area* accommodating men and mine equipment is protected against rock pressure by rows of timber or metal props which are either adjusted directly under the roof of the seam or bear the pressure with the aid of *cap sills* or other elements of timbering lying close to the roof (see below). In the past, this was commonly referred to as "face man's timbering", since it was mostly the face men who installed it. Because of mechanised coal production and labour differentiation in coal teams or gangs, this term has now become obsolete, and *face timbering* is used instead. The latter can support the roof and prevent it from caving in the immediate vicinity of the coal face, where the roof of the seam and the rocks overlying it are held in place chiefly by cohesion with the rocks still resting directly on the solid mass of coal. At some distance from the breast of the face, usually within a few metres and depending upon the properties of rocks, the effect of this cohesion drops to a point where post timbering becomes inadequate.

When the excavated area is gobbed up (Fig. 129, *1*), rock pressure is borne by the mass of mine-fill. Where mining involves caving, a *special timbering* is set up along the break or rib line. It consists of wooden or metal cribs, closely spaced rows of posts (Fig. 129, *2*), steel organ timbering of walls and other types of special timbering (see Section 3).

Special timbering near the face and mine-fill are designed to bear rock pressure and to protect the active stope area from caving. The disposition of timbering at the faces, the dimensions of its elements and the sequence followed in setting it up are depicted in a drawing called "*timbering certificate*".

Under consideration at present and undergoing trial tests is an idea of protecting the stope space with the aid of *movable steel shields*

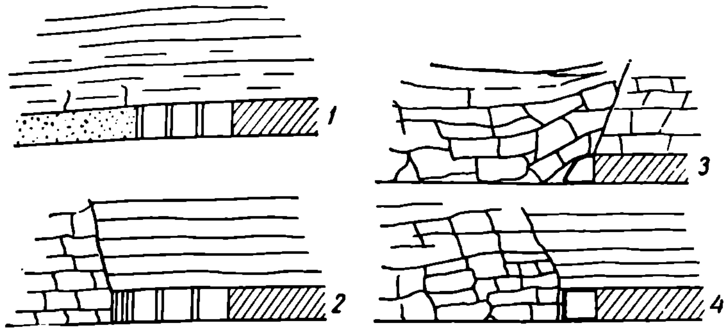



*Fig. 129* A diagram illustrating protection of an active stope area
*1*—by props and fill; *2*—by props and special timbering; *3*—by movable protective shield; *4*—by movable mechanised support timbering

(Fig. 129, *3*), and work is being carried on to introduce *mechanised movable timbering* (powered movable support) (Fig. 129, *4*).

In the case of continuous production faces worked in slightly inclined and sloping seams, control of wall rock pressure actually boils down to control of roof rocks or, in short, to *roof control*.

In order effectively to control the roof one should know the nature of rock pressure. Let us examine this problem in an elementary manner conformably to conditions prevailing in continuous working faces during the mining of gently inclined and sloping coal seams.

In order to *open up* (or *cut up*) a continuous face (longwall), *through-cut 1* is pushed forward over its entire length (Fig. 130). Since the width of this through-cut is insignificant (usually about 2-3 metres), even if its roof should cave in (which is naturally inadmissible) there would be only a small volume of rocks *2* lying under the dome of natural equilibrium which, according to the well-known theory advanced by M. Protodyakonov, forms in such cases over a mine working, that would actually fall. Consequently, according to this viewpoint, the pressure of roof rocks on the timbering of a through-cut cannot exceed the weight of rocks lying under the dome of natural equilibrium. Generally speaking, the stronger the rocks of the roof capping, that is, the greater the cohesion between their particles, the less this pressure is. With strong rocks it is sometimes possible to do without any

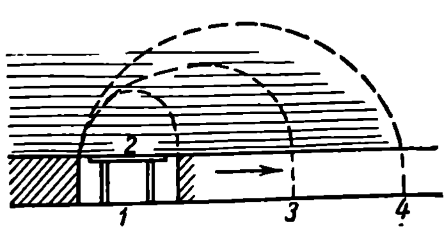



*Fig. 130.* Initiation of mining in a continuous production face

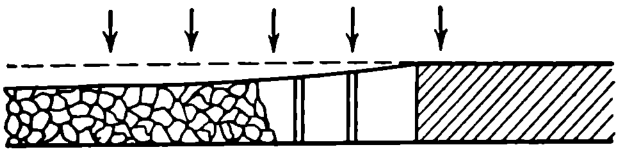


*Fig. 131.* Diagram illustrating the nature of rock pressure bearing on the working face with mine-fill

timbering in the through-cut, for there is practically no rock pressure there.

Consequently, rock pressure bearing on the working face will be low at the very start, and if wall rocks are firm it will not manifest itself at all.

But with the production face advancing into positions *3*, *4*, etc., the span width of the mined-out area will augment, the height of possible natural equilibrium domes will go on increasing, and so will pressure on the face timbering.

Subsequently, the nature of rock pressure to which the timbering near the working face is subjected will vary, this depending on whether it is gob that is packed or the capping of the roof that is allowed to fall.

As said above, roof pressure in mining with fill (Fig. 131) is borne by the latter. But, since the mine-fill shrinks under the pressure, the roof in the area near the face is apt to subside a little. Consequently, the pack does not eliminate rock pressure in a production face, but only reduces its intensity and makes it more uniform.

When roof rocks cave in, rock pressure proves much more complex (Fig. 132). The active face is supported by face and special timbering. Fig. 132 shows special timbering consisting of a double row of closely spaced props—the so-called *organ timbering 1*. It also reveals that immediate roof *3*, right behind the organ

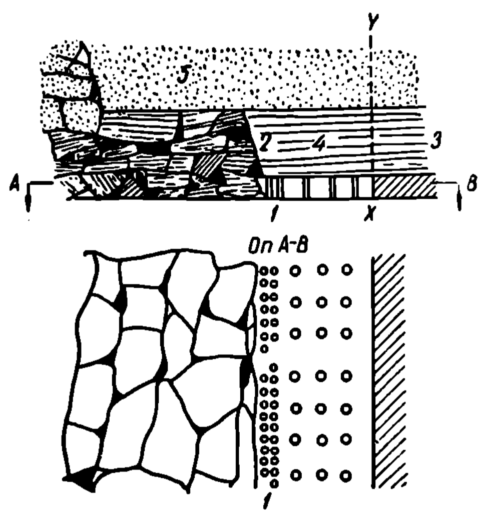



*Fig. 132.* Diagram illustrating the nature of rock pressure bearing on the working face in caving

timbering, has caved in and that *break line 2* runs near the organ timbering. Consequently, hanging over the production face are the still uncaved rocks of the immediate roof in the shape of *back slab 4*, the cross-section of which equals the thickness of rocks constituting the immediate roof and the length is equal to that of the working face. This back slab is held in place, firstly, by the face timbering and, secondly, by the forces of cohesion acting along vertical plane *xy* and involving the rocks that continue to rest directly upon the coal seam (to the right of plane *xy*). Thirdly, there may be some degree of cohesion between the back slab and overlying rocks *5* along the bedding plane. The strength of timbering, the physico-mechanical properties of rocks and the size of the back slab make the relative value of the forces holding this slab over the stoping area vary substantially. At any rate, the wider the back slab, that is, the distance between the rock

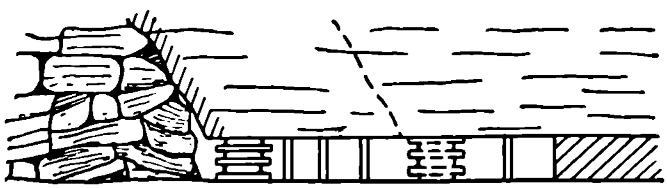



*Fig. 133.* Diagram illustrating roof control by artificial caving

break line and the coal face, the weaker the holding effect of cohesive forces acting along plane *xy* (as in the instance of a cantilevel beam where, all other conditions being equal, the deflection of its end portion is proportional to the length to the beam itself). Therefore, the greater the distance between the production face and the rock break line and the special timbering, the larger, generally speaking, is the rock pressure exercised upon the timbering installed in the active stope face area. To reduce this pressure the roof should be allowed to fall or, in other words, to *cave in*. The idea behind this method is as follows (Fig. 133). When the coal face is moved away from the roof break line and special timbering *1* to a distance where it becomes necessary to proceed with fresh caving of the roof, a new row of special timbering *2* is set up (by the transfer or, at least, partial removal of posts from old row *1*). Between rows *1* and *2* the timbering is *pulled out* (see Section 5, below) and then the roof rocks over this area are allowed to cave in, with the break line running near the new line of the special timbering (dash line in Fig. 133). After the caving, the width of the back slab is reduced and its pressure on the face timbering is weakened.

In this case, roof control is thus effected by *artificial caving.* A more detailed description of this operation is given below, in Section 5.

The distance between each artificial or induced caving of the roof is termed *caving space interval* or *run*. When coal is extracted by cutting machines or combines, this interval or distance is usually determined by the number of cuts. Hence such a commonly used expression as "caving interval after one cut or after three cuts". The more solid the rocks, the bigger the caving interval. Caving after one cut is practised when the roof is poor and, especially, loose.

If the roof is not allowed to fall in proper time, the pressure exerted by the back slab may grow strong enough to destroy the face timbering and may eventually lead to the *spontaneous* caving of the roof over the face area, that is, the "rib" or break line of the roof will run in the immediate vicinity of the coal face. This would mean a breakdown in the production face, a fact that lays particular stress on the necessity for the timely artificial caving of the roof, with its space interval arranged on the basis of previous practical experience and in conformity with the local conditions.

In the initial period of the operation in the wall, that is, when the face has not yet advanced too far from the through-cut (see Fig.130), the roof of the seam rests on coal on two sides.

In these conditions, induced caving of the roof is a rather difficult task and the run or interval of the *first* caving is made larger than normal and, if necessary, blasting is resorted to to induce the roof to settle.

Fig. 132 is illustrative of a particular yet very typical case when an immediate roof, falling directly after artificial caving, is overlaid with thick strata *5* of harder rocks (the immediate roof, for example, includes clay shales, with capping of sandstone or limestone stratum). Hard rocks then do not fall in during induced caving but, like back slabs, hang over the mined-out area. As the stoping advances, these slabs grow bigger until they eventually become subject to spontaneous caving, which, involving large rock masses and spreading over considerable areas, is attended by rock vibrations and sound effects—rumbling and bumps of exfoliating, fracturing and brusquely subsiding rocks. The intensity of these phenomena vary widely, depending on the bedding succession and physico-mechanical properties of the rocks. This periodic spontaneous fall and subsidence of rocks lying directly over the immediate roof is called *secondary caving*.

The overhanging solid rock masses in worked-out spaces may cause *rock bursts*. At the back of this phenomenon lies the fact that coal lumps (as well as other minerals) suddenly break off and fall from the face breasts and the walls of mine workings, the timbering collapses, the coal pillars become crushed and the workings deformed and even destroyed, etc. All this is accompanied by strong sound effects and proceeds extremely rapidly, explosion-like. Rock bursts may be attended by local earthquakes on the surface.

Rock bursts are attributed to the following causes. Rocks in the earth's crust are usually under a constant stress, and this stress increases with depth. When the working seam is topped by a stratum of firm rocks, they sometimes hang over large mined-out areas without falling. But their huge weight passes onto the protective pillars that happen to be in the worked-out area and onto the edges of solid coal masses surrounding it, this giving rise to elevated *bearing pressure*. In the case of strong coal, before its block edges are crushed, they become subject to such high stresses and accumulate so much potential energy of elastic compression that the destruction of coal is accompanied by rock bursts. In the U.S.S.R. they are commonplace in the mines of the Kizel coal fields where coal and country rocks are so much stronger than in our other basins.

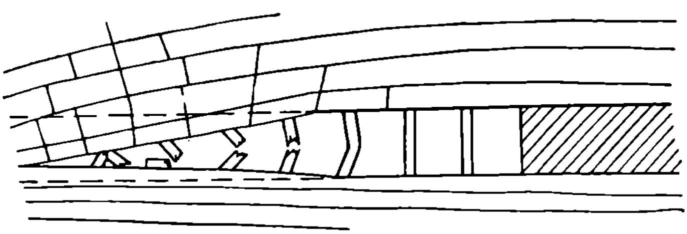


*Fig. 134.* Rock pressure control by gradual settling of the roof

To avoid the hazards of rock bursts, coal pillars should not be left in worked-out areas, and rib fills or pack walls should be built instead (see Section 4, below). Strike entries running ahead of the face should be supported by reinforced timbering over a distance of up to 50 metres. Roof control requires partial fill.

Sometimes, when large rock masses subside suddenly, the secondary caving may be so violent that considerable volumes of air are instantaneously pushed from the goaf into the adjacent mine workings, causing *air blasts* that knock men off their feet or expose them to injuries by flying objects.

Such phenomena are usually preceded by warning signals in the form of rumblings or bumps coming from worked-out spaces. When these signals come, the men should be immediately removed to a safe place.

When a seam is topped by unfirm rocks with more or less same properties, secondary cavings usually do not occur.

Prior to falling, roof rocks as a rule *sag* or *deflect*. Therefore, if the seam is low, the roof may come down to the bottom before it actually starts to cave in (Fig. 134). The convergence of the roof and bottom may also be facilitated by the *heaving* of the latter. In such conditions it may be superfluous to induce the caving of the roof.

Rock pressure control on the basis of this phenomenon is called *gradual roof settling.*

If, along with the gradual settlement of the roof, local conditions allow its periodic caving, the area near the working space must be protected by special timbering.

Of great interest is the relationship between rock pressure in the wall and its length. The following considerations may be set forth in this respect.

As stated above, one of the forces keeping the roof in its position is the cohesion of rocks hanging over the mined-out area along plane $xy$ (see Fig. 132) with those still borne by the unextracted coal block. In this sense, intact coal serves as a *support* for the roof rocks.

If we take circular area *1* (Fig. 135) with its centre in the middle of the face, unextracted coal supporting the roof will occupy half of this area. If we take similar areas on both ends of the face, the supporting coal surface at lower end *2* will occupy 0.75 and at the upper end of face *3*—0.25 of the same circular area. Hence, as compared to the central portion of the wall, the roof pressure near its lower end is smaller and at the upper one—greater. The effect of the specific conditions of support at the ends of the face on the stability of the roof reveals itself over a certain distance along the length of the wall, depending on the firmness of rocks. The weaker and the more susceptible they are to caving, the shorter the distance. For average conditions in coal mines, this distance ranges from 10 to 20 metres. Hence the very important conclusion that it is only in very short walls that the special conditions at the ends of the face can substantially influence rock pressure. In a wall of considerable length—100, 200 or 300 metres—rock pressure will remain uniform everywhere except at its ends. An increase in the volume of work done in a longwall will naturally complicate rock pressure control therein.

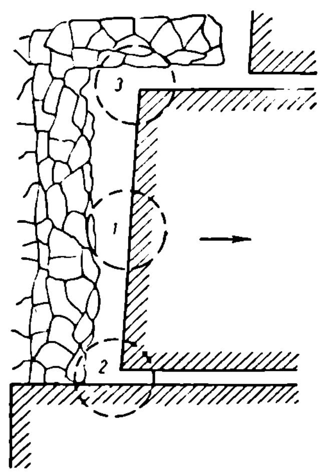



*Fig. 135.* Diagram illustrating the difference between rock pressure in the centre and at the ends of a longwall

The phenomena characterising rock pressure at the production face should be viewed in their *dynamic development.* The advance of a coal face causes roof pressure at a given point of the mined-out area to change. For example, a post that is sufficiently strong while standing at the coal face may break if the distance between it and the face increases.

We have already seen how rock pressure may be reduced by the timely artificial caving of roof rocks. We have also explained why the first artificial caving of the roof is usually more difficult than the subsequent, etc. Therefore, a clear and adequate idea of rock pressure may be gained only if we take due account of the dynamic changes attending these phenomena.

The phenomena of rock pressure appear to have a dynamic nature even if the face remains stationary. Experience shows that in a stationary stope the rocks, as a rule, become *settled*. This means that roof rocks continue slowly to sag, exfoliate and fracture, and fall down in small lumps. In the end, rocks in a stationary stope may become so loose that they will start to cave in. Stoppage of a face is particularly undesirable when the roof is made up of argillaceous rocks, all the more so if there is water. Restarting operations in production stopes that have been inactive for a long time may lead to serious difficulties. Therefore, faces kept in reserve for a long time in conformity with the operational plans of the mine or stopped temporarily for some reason should be periodically *refreshed*, that is, advanced over a short distance so as to eliminate the hazards of rock settling. Hence the rapid advance of working faces, very advantageous in general, is also favoured in the case under discussion. What we have said above also makes it clear that, all other conditions being equal, the higher the rate of face advance the greater the adopted space interval or run of induced roof caving.

Let us now pass to the systematic description of mining operations in a continuous production wall.

## 2. Extraction of Coal

At present coal extraction in continuous faces of thin and medium seams can be fully mechanised by the use of coal cutters, combines, coal ploughs, air hammers, and by blasting.

Normally, *cutting machines* are used to make a *lower* or *toe cut* in the seam (Fig. 136) to facilitate the breaking of coal. The depth of the cut ordinarily ranges between 1.5 and 2.4 metres, the height between 12 and 14 centimetres. In individual instances, when the kerf is made in a soft coal bench or a rock interlayer, the first cut is made in the centre. But this is less effective, for it makes the subsequent breaking of the ground coal band more difficult.

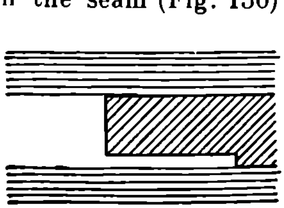


*Fig. 136.* Toe cut

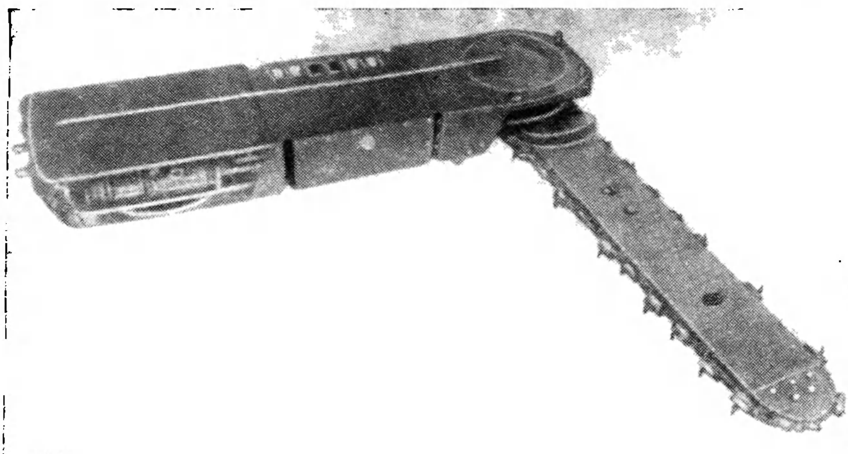


*Fig. 137.* ГТК-35 coal cutter

If, on account of structure features and the physical properties of the seam, coal is apt to fall immediately after being undercut, the cut should be secured by placing posts or *punch props* in it.

The designs of Soviet cutting machines have been modified and improved in the past two decades. Faces of slightly sloping seams are today worked by mining machines provided with bars. It depends on the thickness of the seam and the hardness of the kerf whether 310-mm-high coal band-cutting machines (ГТК-35, ГТК-3м) or the more powerful ones КПМ-2 and МВ-60, 375-400 mm high, are used.

Coal cutter ГТК-35 (Gorlovka heavy-duty, rope coal cutters with motor capacity of 35 kw, Fig. 137) makes it possible to change the cutter feed rate in the course of operation within the range of 0.2-0.4-0.6-0.8 m/min, has a drawing pull of up to 5 tons and is provided with a spiral gummer. The overall dimensions of the machine are: length—2.7 metres, width—0.7 metre, height—310 mm; weight—2.6 tons. Feed rate of the machine during manipulations is 12 m/min.

Our mines make a wide use of the powerful КМП (Kopeisk powerful pulsing feed, models 1 and 2) (Fig. 138), coal cutter, manufactured by the Kirov Mine Engineering Works at Kopeisk in the Urals. It is furnished with the so-called "pulsing" mechanism to change the feed rate in the course of operation. Its principal characteristics are: continuous per hour motor power output—47 kw, the range of operational feed rates—0-0.86 m/min, travel speed during manipulations—8.6 m/min, draw pull of 5 tons when operating at full capacity, length of the bar—1.6-2.8 metres, trim-chain speed rate—1.07-1.12

m/sec; overall dimensions: length (gummer)—3.14 metres, width—850 mm, height—375 mm; weight—3.3 tons.

Another powerful coal cutter (MB-60) with a 60- or 65-kw motor is at present used in making starting cuts in seams with hard coal, as well as a ground structure for coal mine combines.

The MB-60 has a device capable of changing feed rates in the running machine within the range of 0.27-0.54-0.81-1.08 m/min, travel speed during manipulations—14.5 m/min, length of the bar—2-2.8 metres, trim chain speed rate—1.9 m/sec. The draw pull at full load—7 tons. The machine is provided with a spiral gummer. The overall length—3.13 metres (without gummer), width—740 mm, height—400 mm; weight—3.5 tons.

Many coal-cutter operators in the Donets, Kuznetsk and other coal fields have raised their monthly productivity to 15,000-18,000 tons of coal per machine. The record is held by G. Lyashenko of the Zhdanov Mine in the Karaganda basin. Working in a coal wall in accordance with a cyclic operation schedule, Lyashenko attained the unheard-of production figure of 32,000 tons of coal a month. Coal-cutter operators S. Tomashevsky and I. Mozgovoi raised the monthly output of coal per one coal-cutting machine MB-60 at the Lutugin Mine in the Donets basin to 20,000-23,000 tons.

The machines described above are normally used to make a toe cut. For central cuts they have to be set on a frame made of angle and channel bars.

As a rule, undercut coal is broken with the aid of small charges of explosives (usually two-three permissible cartridges). More often than not holes are bored by electric drills (augers). Only soft grades of coal are broken with the aid of pneumatic coal hammers.

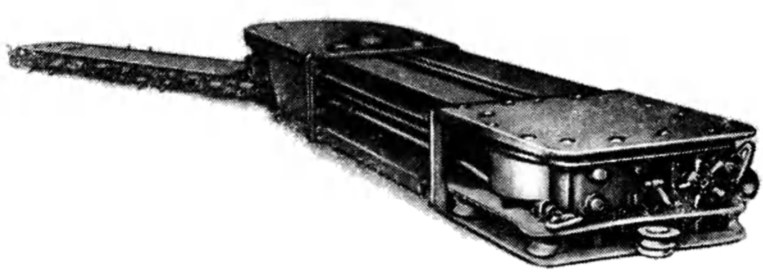


*Fig. 138.* КПМ-2 coal cutter

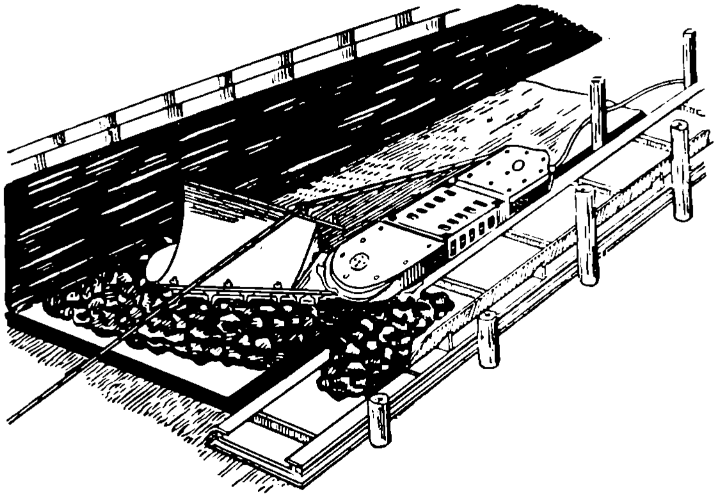


Fig. 139. Cutter-loader

The facemen engaged in breaking coal shovel it onto conveyers. Output per faceman shift varies approximately from 3-4 to 15 tons, depending on the thickness of a given seam and the hardness of coal.

Thus, mechanisation in a working face using a coal cutter is at a very high level and coal is transported with the aid of a conveyer. However, coal is broken and loaded in this case by facemen by hand.

This has raised the question of mechanising coal-loading operations.

The simplest way of mechanising the operation of loading coal onto a conveyer in working walls of gently sloping seams is to use a *cutting-loading* machine (ВПМ). This is an ordinary coal cutter, though equipped with an additional arrangement for loading coal (Fig. 139) in the form of a detachable share or mouldboard, that is, a bent steel sheet that can be bolted to the bar. The cutter-loader operates in conjunction with a rigidly built drag-link conveyer (СТР-30). The cutting-loading machine can move alongside the conveyer (Fig. 139) or directly on it. In the latter case it is somewhat tilted. Since the machine is suitable only for loading loose coal, coal excavation in a wall proceeds in the following manner: after a cut is made, coal at the face is blasted, with about one-fourth of the coal undercut being thrown spontaneously onto the conveyer. About half of the coal is loaded onto the conveyer with the aid of the above-mentioned mouldboard. The remaining one-fourth must be shovelled onto the conveyer by hand, this being the principal disadvantage of the machine discussed. The thickness of the seam should not be less than 0.9 metre.

This naturally has stimulated invention and construction of *coal-mining combines*, that is, of units capable of undercutting, breaking and loading of coal onto conveyers by a single mechanised operation. The U.S.S.R. is first in the world in the invention, construction and practical utilisation of coal combines. The first combine for longwalls, designed by Bakhmutsky, was built in 1934.

Since in conditions prevailing at working faces coal can be loaded mechanically only if it is broken into relatively small pieces, the

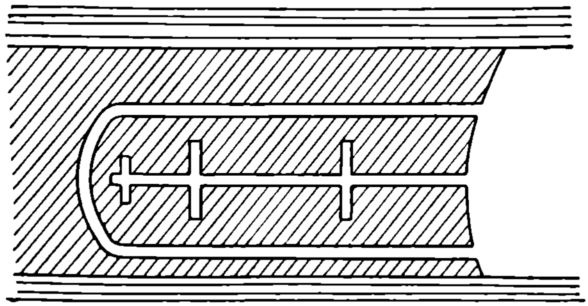


*Fig. 140.* Cuts made by the Donbas coal combine

cuts made by combines must be of a more complex shape. For instance, the Donbas combine makes a looplike cut by its trim chains and inside this cut there is another, made by the cutting bar and toothed disks fitted on it (Fig. 140). Because of this the amount of fines made by combines in the process of undercutting and breaking coal is greater than when coal cutters are used, and this is a shortcoming of the types of the combines currently in use.

The Donbas coal-mining combine (Figs 141, 142), designed by the State Prize winners engineers A. Sukach, M. Gorshkov and V. Khorin, has gained wide popularity in recent years. The operating part of the combine comprises a circular bar with a trim chain and a rod with cutting disks, which make vertical cuts to split the coal band delimited by the circular bar cut.

Coal dust or gum carried by the trim chain of the bar is accumulated in the housing of the gummer, which then loads it onto the face conveyer. The length of the bar and the rod that determines the width of the coal band extracted by the combine varies from 1.4 to 2 metres. Circular bars are manufactured 710, 830 and 1,000 mm in height and are mounted on the combine in conformity with the thickness of the seam. In seams more than 1.4 metres thick, an articulated collapsible

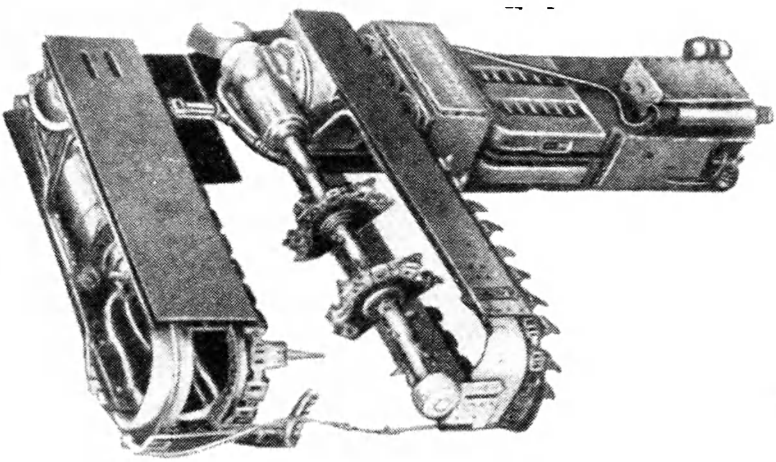


*Fig 141* Donbas coal combine

*Fig 142* Donbas coal combine at a coal face

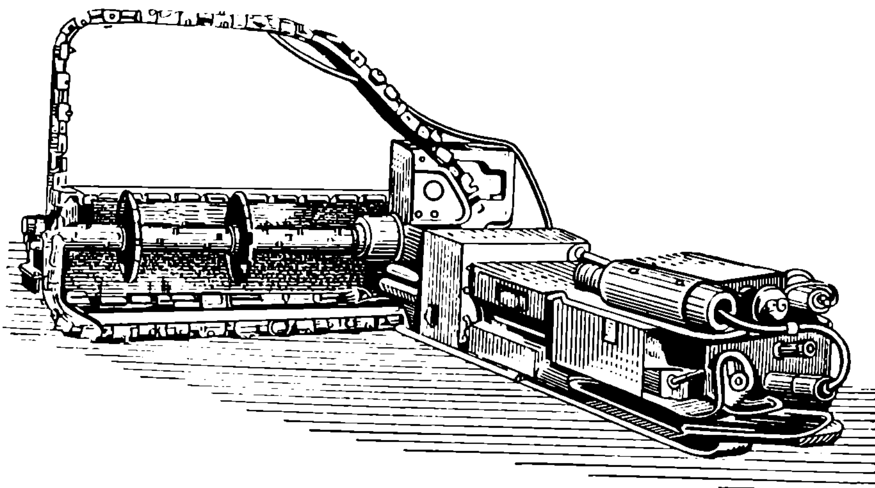


*Fig. 143.* Articulated collapsible bar

contour bar is used, as proposed by the staff of the Kirov Mine in the Kuznetsk coal basin (Fig. 143). The height of the bar in the operating position ranges from 1.3 up to 1.65 metres, depending upon the size of the interchangeable insert in its vertical arm, with the grab (cut) 1.6 metres wide. The trim chain speed rate in all the bars is 2.14 m/sec.

A circular scraper loader, its chain driven by an independent 13-kw motor, is mounted at some distance from the bar and the cutter-breaking rod.

In Karaganda, slice mining at the "Verkhnaya Marianna" seam is successfully done by twin (coupled) Donbas combines, which extract slices 2.5-3.0 metres thick. The loaders of the twin combine are powered by a 35-kw motor.

The main electric motor of the combine, which drives the operating mechanism and the feeding arrangement, has a capacity of 65 kw. The feeding rate varies stepwise, depending upon the hardness of coal, within the range of 0.27-0.54-0.81-1.08 m/min.; the idle travel speed of the combine is constant—14.5 m/min. The overall dimensions of the Donbas-1 combine are: length in the operating position—4.6 metres; width—0.72 metre and in the position of idle travel—0.86 metre.

To prevent the formation of gum and dust, the combine is furnished with a spraying device consisting of five to seven pulverisers located in places where dust accumulates most. Water is supplied under a head of 4-5 atm to the pulverising nozzles by a pump with a capacity of 20 litres per minute (powered by a 4.2-kw motor), placed in an entry, through a 16-mm bore flexible hose and a filter and piping on the combine. The consumption rate of 3-4 litres/min per pulveriser is sufficient to reduce the dust level in the face near the combine 6- to 10-fold and thus substantially improve hygienic conditions there.

The Donbas combine is designed to mechanise the mining of soft and semihard coal in seams not less than 0.8 metre thick.

There are several improved types of the Donbas-1 combine put out for mining seams varying in thickness from 0.8 to 2.5 metres, with the range of 1.2, 1.6 and 2 metres.

Fig. 142 illustrates how the Donbas-1 combine is operated in face with metal posts.

The Donbas combine is a highly efficient machine. At No. 31 mine of the Karaganda coal field, operators F. Bushchinsky and V. Velichko produced on the average 24,690 tons of coal a month in 1955, raising the output to a new height of 27,031 tons in December of that year. At that time this was a record.

In 1954 the Kirov Machine-Building Works in Gorlovka put out the new Donbas-2 combine. The capacity of its main motor was 120 kw and that of the loader—35 kw, the operating voltage amounting to 660. Tested in one of the hard-anthracite seams of No. 13 Ayutinskaya Mine of the Rostov coal fields, the combine showed it could develop a high efficiency, while the Donbas-1, in similar conditions, proved to be totally unsuitable. The Donbas-2 combine has been put into serial production.

By its general construction, the Gornyak combine (Fig. 144) is analogous to the Donbas model, but it is intended chiefly for mining seams 0.6-0.85 metre thick containing soft and semihard coal

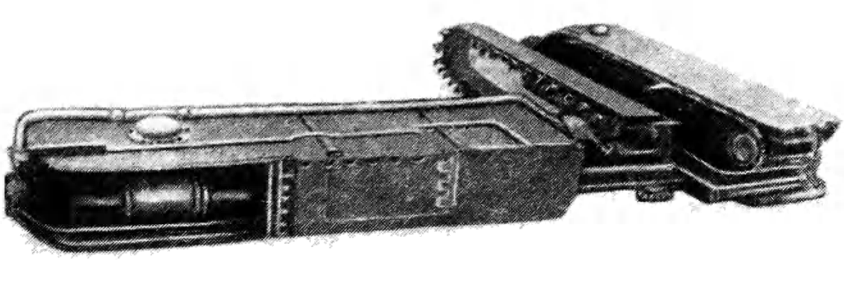


*Fig. 144.* Gornyak coal combine

and may also be used for hard coal seams, provided the coal contained therein is subject to delamination. The design of the machine is somewhat simpler. Its overall dimensions: length in the operating position—4.9 metres; width—750 mm; height—400 mm; weight—about 7 tons.

The Shakhtyor coal combine is also similar to the Donbas, but it can operate in low (0.5-0.75 metre) seams. It makes but one loop-like cut. Crushed coal and gum are carried out of the opening slot by the lower branch of the trim chain. Loosened coal is either disintegrated during the operation of the trim chains or broken into small pieces by the scrapers of the gum loader. Broken coal and coal dust are loaded onto a conveyer by the trim chain and gumstower. The overall length of the Shakhtyor coal combine is 3.8 metres, its width—0.76 metre, gross weight—3.5 tons, motor capacity with continuous rating—47 kw, output—up to 45 tons per hour.

In 1951 a State Prize was awarded to a group of designers headed by A. Gridin for inventing and introducing the УКТ-1 coal-mining combine (for thin seams), in production faces of low, slightly sloping coal beds 0.45-0.7 metre thick (Fig. 145).

The operating mechanism of the УКТ combine consists of four bits *1* with blades *2* and jumpers. The blades separate coal from the face surface without making any starting slots, thus reducing power consumed in loosening and breaking it. Behind the bits, in the operating mechanism guides, runs trim and gathering chain *3* provided with teeth and blades attached to individual cams which, when the chain moves, grab coal and load it onto a face conveyer.

The chain is driven by a sprocket fixed to the spindle of the last bit, and its lower branch moves from the face towards the conveyer, thus ensuring the loading of loosened coal by the bit blades, while some of it is brought onto the conveyer by the bits themselves.

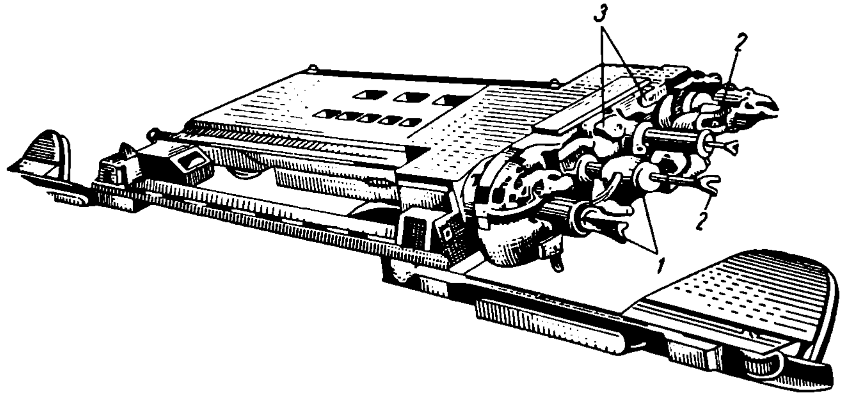



*Fig. 145.* УКТ coal combine

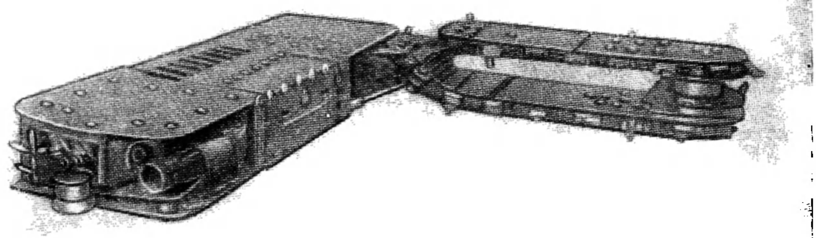


*Fig. 146.* УКМГ coal combine

As required by the thickness of the seam, the machine is furnished with a corresponding set of operating mechanism parts, thus securing a coal-breaking height of 0.45-0.5-0.6 or 0.65 metre. Each set of bits allows the coal-breaking height to vary within a range of 50 mm with the aid of the teeth and the trim and gathering chain.

The feeding and power units in the combine are those from the КМП-1 or ГТК-35 coal-cutting machines. In the first case the continuous rating (per hour) of the motor is 47 kw, in the second—35 kw. The feed rates of the running machines vary correspondingly—in the first instance infinitely within the range of 0 to 0.86 m/min, in the second—stepwise in the sequence of 0.2-0.4-0.6-0.8 m/min.

The width of a coal strip drawn by the УКТ-1 combine is 1.4 metres. The new УКТ-2 model is designed to draw a strip 1.65 metres wide. The angle of the seam pitch may be as great as 20-23°. The travel of the machine within the area of the strip under extraction appreciably reduces the width of the working space, though a roof surface of up to 10 sq m remains unsupported immediately over the combine. The overall operating dimensions of the machine; length—3.1 metres, width—1.6 metres, height—0.45-0.65 metre, weight—4-4.12 tons.

In 1952 the Gorlovka Machine-Building Works put out a pilot and in 1953 a serial lot of two УКМГ coal-mining combine models (coal combine for low seams at the Gorlovka Mine) (Fig. 146), intended for working thin coal seams. The operating mechanism in this machine is constituted by two cutting bars, placed one above the other with a cutting disk in-between. A double-articulated link chain of the lower bar grabs loosened coal and gum when moving in the direction opposite to that of the trim chain in ordinary coal cutters, and loads them onto a face conveyer.

When the range is 1.45-1.65 metres deep, the feeding rates in a running machine vary from 0.2 to 0.8 m/min and the continuous (per hour) rating of the electric motor equals 35 kw, the output of the combine is from 10 to 54 tons/hour, depending on the thickness of the seam.

The overall dimensions of the machine in the operating position: length—3.3 metres, width—0.72 metre, height—0.31 metre, weight—3.3 tons. The combine may be operated in a small unsupported area and is employed in seams of irregular thickness, from 0.38 metre up.

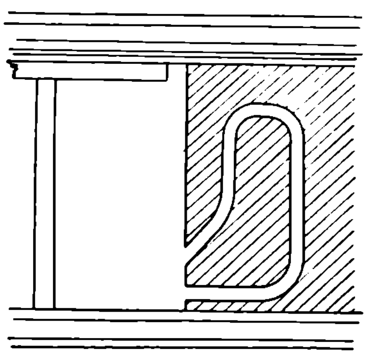


*Fig. 147.* Cut made by the BOM cutter-breaker

Original in design is the operating mechanism of the BOM-2 mine combine (cutter-breaker, model 2) built by State Institute for Designing Coal Equipment (Fig. 147). The position of the BOM combine in the wall is seen in Fig. 148. It makes vertical cutting slots of up to 2.8 metres in height and 130-140 mm in width. Coal bands 0.3 metre thick, formed between the cuts, are loosened up and the coal lumps are loaded onto a face conveyer by a mouldboard or a circular flight loader (in the latest model of the BOM-2M combine). The machine cuts coal in strips 0.9 metre wide and is used in soft and semihard coal seams not less than 1.5 metres thick, chiefly in the Moscow coal fields.

Mine combines may also be employed in sloping seams, but their employment in such cases is distinguished by a number of specific features (Fig. 149). To prevent it from falling when pull rope *5* is ruptured, the combine is tied by safety rope *4* to drum *6* of special hoist *7*. Broken coal slides down immediately and for that reason the machine has no loader and there is no conveyer in the face. The functions of the other parts of the unit are explained in Fig. 149.

On the suggestion of the State Institute for Designing Coal Equipment, a new method of mechanised coal drawing with the aid of coal ploughs was tried in continuous walls after the Great Patriotic War. Moving along the face, the coal plough cuts coal 0.6-0.7 metre high and 20-25 cm thick.

The set of mechanical equipment in a coal-plough-worked wall (Fig. 150) includes: coal plough *1*—a heavy-duty (weighing about 3 tons) steel casting in the shape of a share with cutters of special

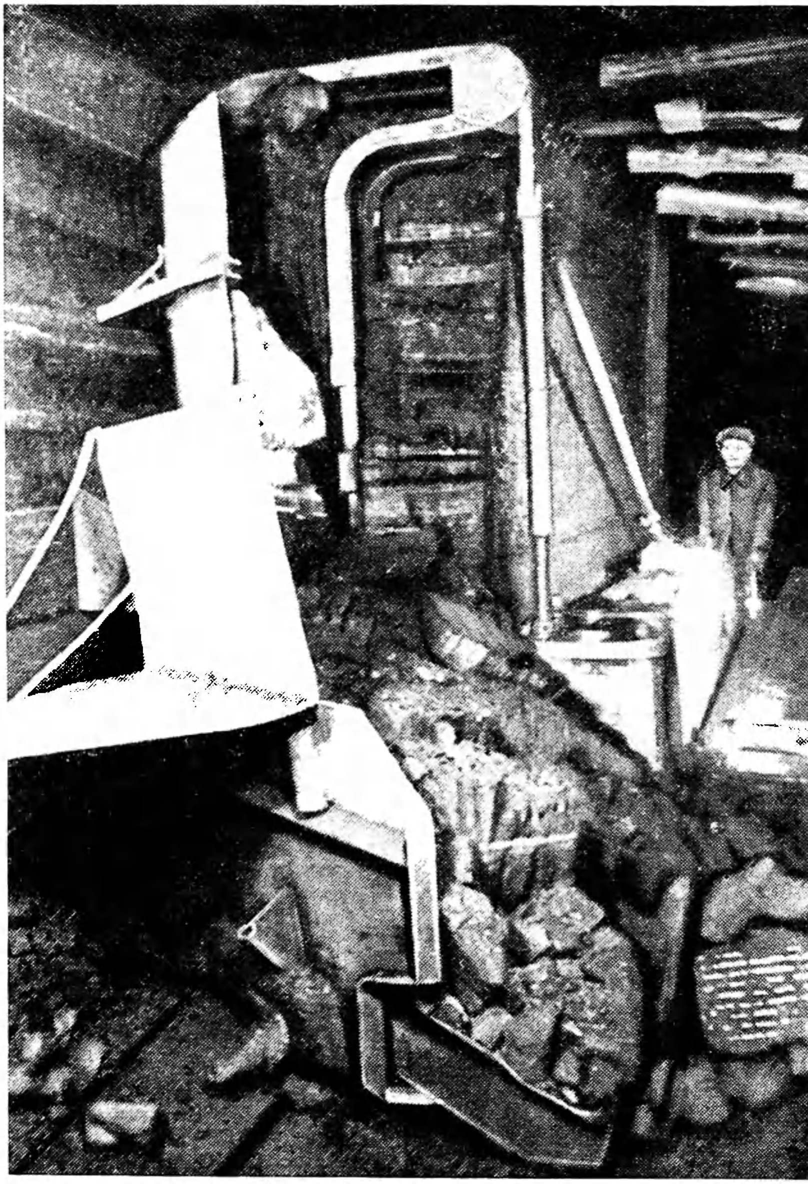


*Fig. 148.* BOM coal combine

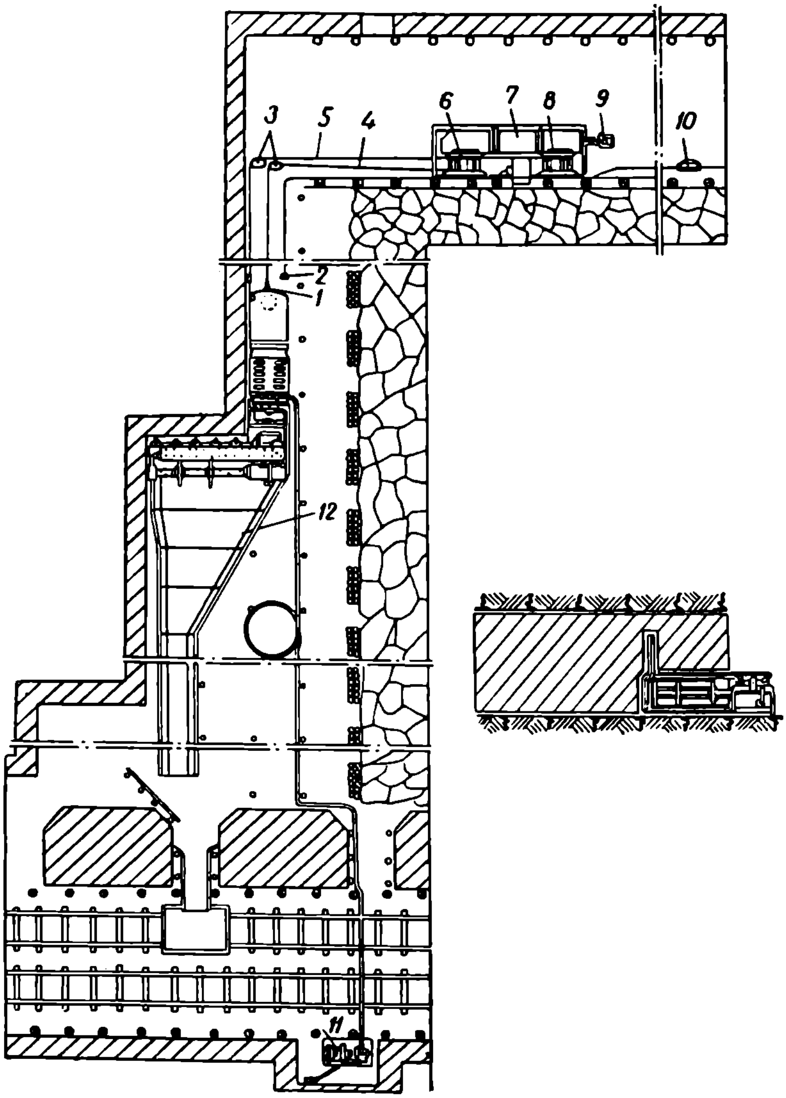



*Fig. 149.* Combine operating in a sloping seam wall

1—coupling device; 2—control desk; 3—movable corner post; 4—safety rope; 5—load line; 6—safety rope drum; 7—hoist; 8—load line drum; 9—support for securing the hoist; 10—magnetic starter; 11—spraying pump; 12—face tray (left)

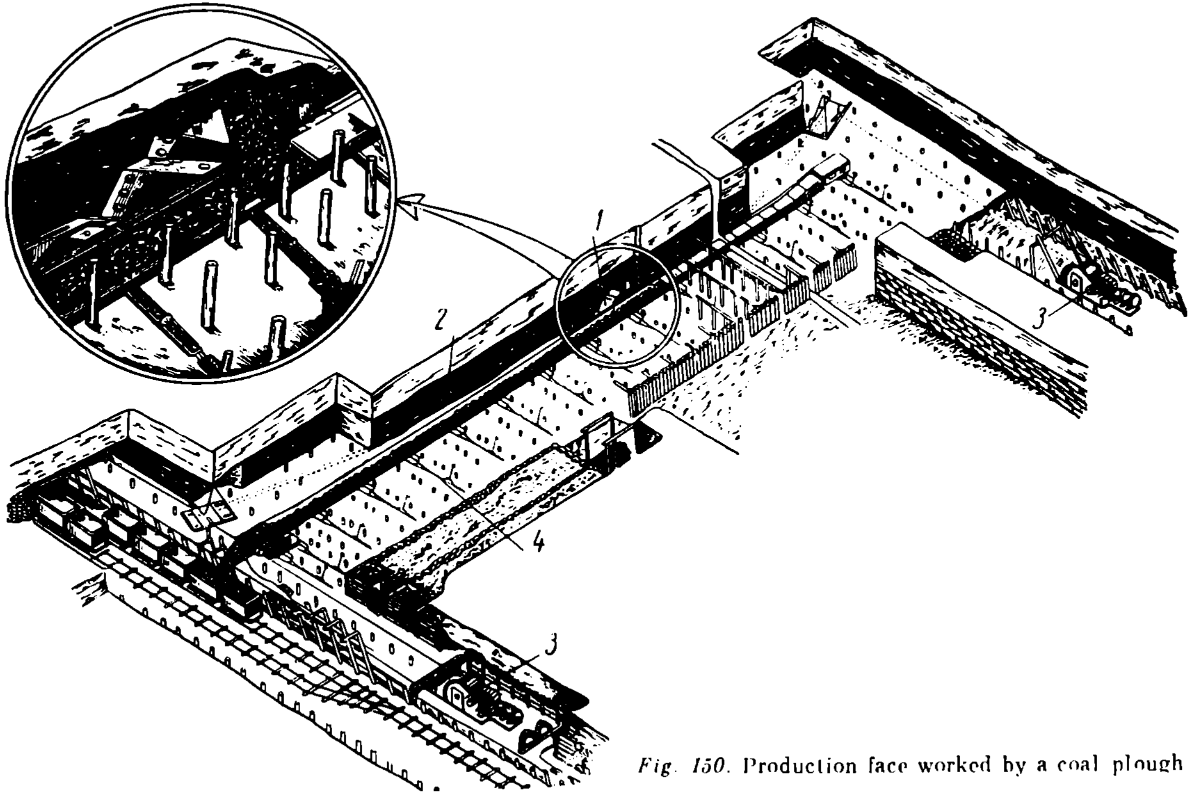



*Fig. 150.* Production face worked by a coal plough

steel inserted into it for loosening coal; flight conveyer *2*, which is used not only to transport coal but also to steer the plough; two slow-speed hoists *3* set up in entries to pull the ropes propelling the plough; and pneumatic jacks *4* to facilitate shifting conveyers to the face. The coal plough is supplied with mouldboards to load coal onto a conveyer, and moves along the wall at a speed of 6-8 m/min. After each run of the plough the conveyer is shifted to the face by jacks thrust against the supporting metal posts.

Field trials have shown that these coal ploughs are capable of operating in very soft coal only, in seams whose occurrence is uniform and roof and bottom rocks sufficiently stable and firm. With the width of the coal plough equalling 0.43 metre, that of conveyer—0.7 metre and the space between timber rows—1 metre, the untimbered portion of the active stope area is up to 2.1 metres wide, and that is inadmissible in the case of rocks of medium stability. This leads to difficulties in roof control, and the time lost on account of the frequent shiftings of the conveyer makes it impossible to produce large quantities of coal. That is why coal ploughs of this type have as yet found very little practical application.

In recent years, however, *high-speed* coal ploughs have been utilised on an increasing scale.

In the U.S.S.R., a combined coal plough of this type has been proposed in the Kuznetsk coal fields by the Stazhevsky brothers

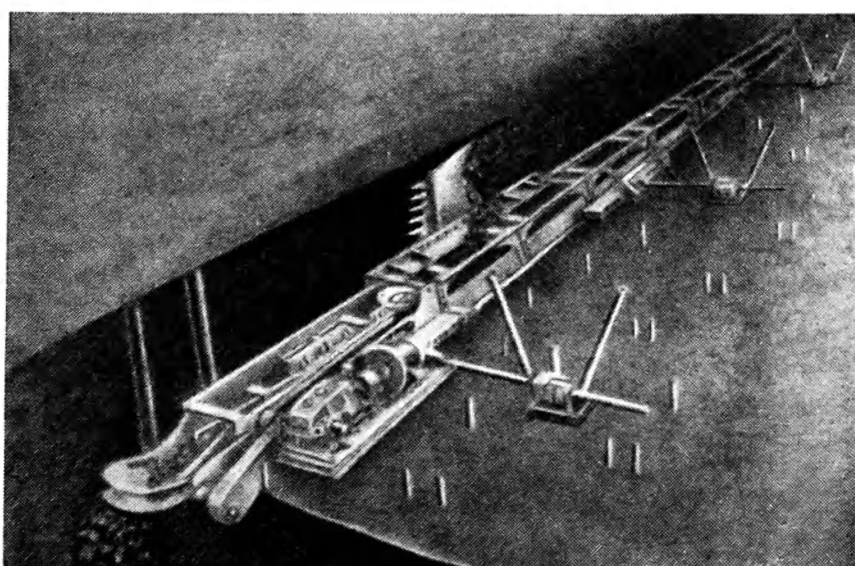


***Fig. 151.*** **Coal plough designed by the Stazhevsky brothers**

(Fig. 151), while abroad wide use is made of Westfalia-Lünen coal ploughs (Fig. 152).

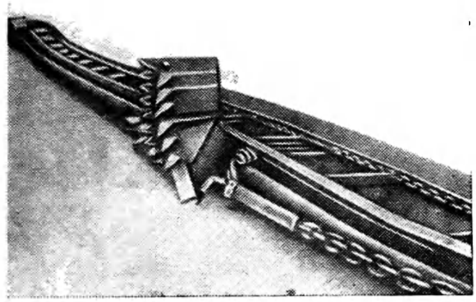


*Fig. 152.* Westfalia-Lünen coal plough

In high-speed coal ploughs, the cutting mechanism is not fitted with smooth blades, but with massive teeth welded on with a hard alloy and used for dislodging coal from the surface of a face by cuts 50-100 mm deep. Moving along the face with a speed of 30-45 m/min, the plough removes coal strips almost over the entire thickness of the bed with the draw pull under load of up to 7-10 tons.

Both the above-mentioned and the high-speed coal ploughs cut coal by pressing their teeth or blades against the breast of the face under the effect of draw pull transmitted through a rope or chain. It is common, therefore, to designate these ploughs as *static*.

After each cut the whole unit is moved nearer to the face breast by special mechanisms which, in the home-made KC-2 plough, are actuated through the idle running line passed over the by-pass rollers at the end sections of the plough unit conveyer.

During the test-trials of the KC-1 and KC-2 strippers at a seam with sufficiently viscid coal at the Kirov Mine in the Kuznetsk coal fields the face advance in individual shifts was as high as 1.8-2.3 metres.

Coal literature reveals that in May 1954 there were 72 coal ploughs working in West-German mines, their daily output being 30,000 tons. In England, at the end of 1953, there were high-speed ploughs in operation in nine longwalls and ordinary static ploughs in ten.

The tendency to extend the field of coal plough application to viscid and harder coals has resulted in the construction of *percussive* (thrust) and *vibration* action coal ploughs. Such strippers were tested both in our and German and English mines, but as there is insufficient field experience it is as yet premature to speak of any extensive application of this type of equipment.

Seams with soft coal can be worked with coal hammers. In practical use today are only air hammers, since the persistent attempts (of Prof. Shmargunov and other inventors) to design and utilise *electric* coal hammers in coal mines have so far remained unsuccessful.

## 3. Working Face Timbering

It is above all from the viewpoint of *safety* that the timbering of working faces fully conforming to the pressure and properties of rocks acquires particular importance. An analysis of the causes underlying accidents in mines shows that a large proportion of them are due to the falling lumps or blocks of the useful mineral or barren rocks. Besides, since cavings occur over considerable areas, a production face may become a scene of mass accidents.

Special safety measures should be taken in the case of rocks split by cleat planes, fissures or made of blocks which detach easily from the solid mass in places where the surfaces are glossy and slippery. Posts, cap or head boards and laggings of timber sets should be placed so as to eliminate any possibility of roof falls. Particular attention should be attached to sites marked by geological disturbances, where rocks are less rigid than is usually the case. One safeguard against mass cavings in work places is adequate roof control. Working at the face, the gang leader and the miners, as well as the technical supervising staff, should be on a lookout for any signs of impending caving in the roof and on the face breast, check up on the firmness of the roof by tapping it and, when necessary, set up additional, reinforcing timbering.

Since huge and ever-increasing amounts of coal are mined in the U.S.S.R., the use of timber sets at production faces represents an extremely big drain on timber resources. For this reason metal sets are used on an ever wider scale both in production and development mine workings.

Let us first discuss the timber and then the metal supports of working faces.

The space immediately adjoining a working face is timbered by •
*posts* or *props*.

Posts are placed along the wall in regular rows, this being absolutely indispensable when coal cutters, mine combines and conveyers are operated at the face.

It is only when the roof is very firm that the ends of the posts can be blocked directly against the roof. Almost invariably, however, practice calls for putting *head boards* made of wood slabs, sawed lumber and even round timbers, between the posts and the roof, this depending on the properties of and pressure by rocks.

The cap or head boards may be placed along the wall (Fig. 153) or perpendicular to it (Fig. 154).

In the first case, slabs used for cap boards overlap each other and are supported by a common prop at their ends. When timber is heavier, sawed or round, the head boards meet end to end and the ends are supported by extra posts.

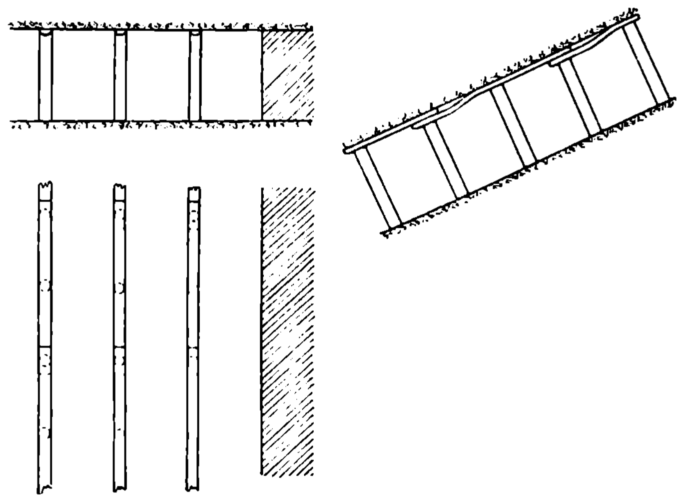


*Fig. 153.* Timbering with head boards running along the face

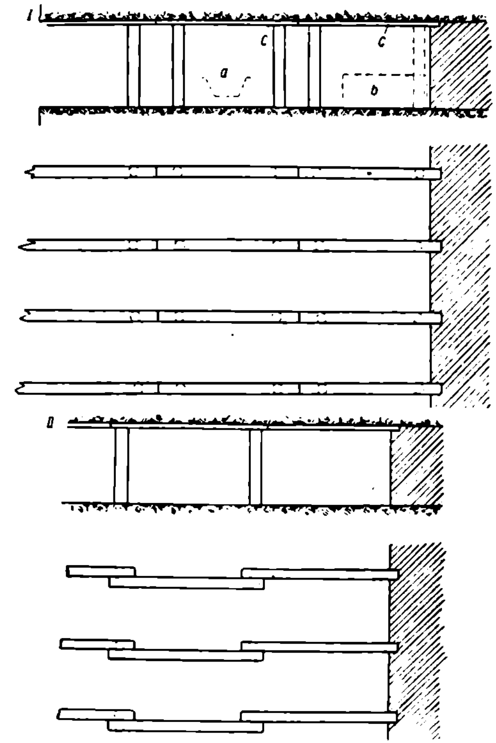



*Fig. 154.* Timbering with head boards placed perpendicularly to the face

Disposition of head boards along the wall has a number of substantial disadvantages:

1) Development of rock pressure within the area of an active face usually causes fissures along it. In order to prevent rocks from breaking away as the result of that, the head boards should not be placed parallel but across these fissures.

2) The part of the face area immediately adjoining it is devoid of all support.

3) The spacing of timber rows, though conforming to the size of equipment used at the face, that is, coal cutters, mine combines, conveyers, etc., may at the same time be undesirable from the standpoint of roof stability.

For this reason it is a much more frequent practice to place the head boards perpendicular to the face (Fig. 154). Two posts are usually wedged under and against the head board (the so-called frame sets). The width of the frame (*I*) should be sufficient to permit setting up conveyer *a*. When coal cutter *b* passes at a given point of the face head board end *c*, adjacent to the face breast, is forced into coal. After that a prop is set up right next to the coal surface. Sometimes these frames are arranged in a somewhat different way (*II*).

The size of timber and the number of posts set up on the average at intervals of one sq metre depend on the thickness of the seam and rock pressure. The props are usually 10-20 cm thick. As stated above, the distance between posts in a frame set should be sufficient to allow the installation of a conveyer. When coal cutters are used at the face, the disposition of the timbering should be brought into line with the depth of the cut.

The distance between frame sets is determined by the firmness of rocks: the weaker the rocks the closer the spacing of the sets.

When the roof is so weak that rock pieces fall down through the interstices of head boards, additional laggings (thin slabs) are placed against the roof, their ends over the top of head boards.

The ordinary face timbering of longwalls serviced by conveyers in slightly sloping seams, described above, is inconvenient because the conveyer has to be dismantled (unbolted) before it is shifted to a new position. This disadvantage may be eliminated by the use of the following type of timbering (Fig. 155). The head boards should be sufficiently long to allow posts *1* and *2* to be blocked under both ends of head boards when the conveyer is in position $k_1$. Before the conveyer is placed in the next position, $k_2$, new head boards *b* are put in place, their ends extending far beyond the ends of the first head board set, *a*. After this posts *3* are set up and posts *2* are pulled out, this allowing sufficient room for the transfer of the conveyer which then can be shifted to a new position without being unbolted. Therein lies the main feature of the method under consideration. But it can be used only in the case of very rigid rocks, which make it possible to set up widely spaced timberings and regularly shift and reset the props.

In Section 1 of this chapter we learned that caving methods in mining involve setting up *special* timbering in addition to that put up directly at the coal face. Here we shall dwell on some details.

A timber *crib* consists of sticks arranged as shown in Fig. 156. As a rule, cribs are made by laying props similar to those used in supporting working faces, though sometimes other types of timber

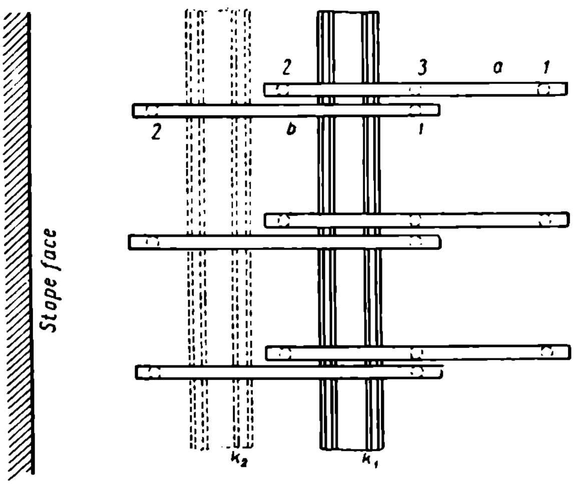



*Fig. 155.* Face-timber allowing a conveyer to be shifted

are also employed (very often partly broken timber obtained when repairing development openings). Ordinarily, a crib is laid near the earlier installed props of the face timbering. To make it stable, the crib is wedged, but the wedges should be driven between the posts and not between the crib and the roof, since in the latter case the contact surface between the crib and the roof would be smaller, and this would weaken the roof support. As will be seen later, in most cases cribs have to be shifted from place to place, and to facilitate their dismantling they are underlaid with pieces of rock. To make the cribs fall apart, these rocks are broken.

Cribs are arranged in single or double rows along the working face, usually at intervals of about 1-2 metres and more (up to 7 metres), depending upon the pressure and stability of the roof and the thickness of the seam.

Where induced caving is practised, the space between crib runs is determined by the caving interval or rate. When no caving is induced by pulling out timber, spacing is determined purely empirically, on the basis of accumulated experience. More often than not the intervals range between 2 and 8 metres. It is only natural that in one and the same wall the space between cribs varies along with the operating conditions. The necessity of setting up a new row of cribs comes with the increase in roof pressure.

In most instances, the building of a new crib row implies simultaneous demolition of the old ones, and that is the reason why cribs are shifted. This is usually started with the top cribs.

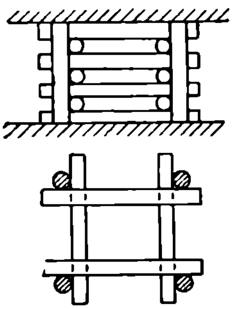


*Fig. 156.* Crib

In addition to systematic cribbing, there are extra individual cribs set up, as the need arises, in places where, for one reason or another, roof pressure increases abnormally.

The above-mentioned close-set rows of posts are called *organ timbering.*

The posts usually have to be arranged in double rows, less frequently in single or triple rows. In these the posts almost touch each other. Where pulling out or knocking out of timber is practised, passageways for men, not less than 0.8 metre wide, are left in the organ timbering at intervals of not more than five metres. In seams about one metre thick one worker sets and blocks in place between 80 and 120 organ-timbering posts in a shift.

Very often the practice provides for a limited use of organ timbering without cribs. Sometimes groups of closely spaced posts (*clusters*) are set up instead of cribs.

Preference is shown to organ timbering and post clusters because post timbering is *rigid* compared to cribbing which, to some degree, is *yieldable*, compressing and decreasing in height when pressed upon by the weight of the roof because the timber sticks piled one on top of the other crumple up where their points meet. Rigidity is a positive property that helps to obtain a good break line in rocks when the roof caves in.

On the other hand, cribs withstand side pressures better than the posts. To make cribs more rigid, they are built of timber sticks slightly bevelled on both sides or even of heavy planks.

Prior to proceeding with artificial caving, special timbering is set as close to the face as possible, but with sufficient room to allow the passage of a mine combine or coal cutter and the installation of a conveyer after the roof has been settled. The posts and cribs should be arranged in a straight line along the face, this being of importance not only for facilitating the mining of a longwall with the aid of mine

*Sections and plans for mine timbering set-ups in stoping operation*

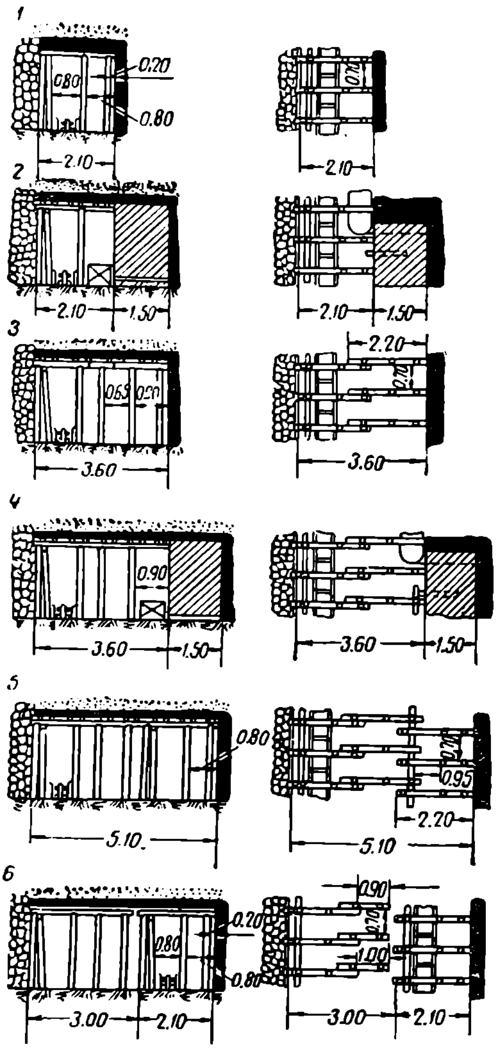



*Fig 157.* Coal-face support in a mine of the Moscow coal fields

combines, coal cutters and especially conveyers, but also for obtaining an adequate break line in the roof when it caves in.

Wall rocks in the coal seams in the Moscow basin are distinguished by their low mechanical strength, and because of that the working faces are subject to high rock pressure and require strong support.

The timbering in the area near the face is made of a row of frame sets with caps running perpendicular to the coal face.

Fig. 157 depicts a typical example of a timbering method employed for supporting the area near the working face in the Moscow coal fields. It shows six successive positions in an interval between two

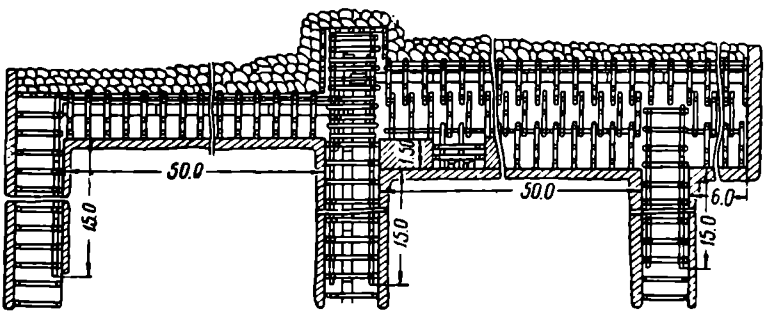



*Fig. 158.* Junction timbering between wall and entry

roof cavings: *1*) position after caving; *2*) at the time the first cut is made; *3*) after drawing coal from the first cut; *4*) at the time of the second cut; *5*) after drawing coal from the second cut; *6*) after shifting the conveyer. Timbering may be reinforced by angle braces, as required by the nature of rock pressure, while coal at the face breast is held in place by vertical boards and stulls. The figure reveals that the roof is caved after every two successive cuts.

Fig. 158 is illustrative of a method used to protect a wall entry by additional organ timbering in the mined-out area.

The use of wood for supporting working faces consumes scores of cubic metres of timber per 1,000 tons of coal produced, and that, considering the huge amounts of coal mined annually, has become an economic problem of national importance and led to attempts at employing *metal* mine supports. These naturally have to be *transferable* or *movable* to allow their multiple utilisation.

Soviet inventors have now designed and tested quite a number of types of metal support in the mines.

The design of *metal posts* should make it easy to shift them from one place to another. Many constructional types of such posts have been proposed.

Widely used in recent years were the СГК posts (mine posts of the wedge type), designed by the State Institute for Designing Coal Equipment. In 1955, however, model M metal posts were put in serial production, and they are lighter and easier to manufacture (Table 11).

*Table 11*

**Metal Post M**

| Characteristics | Types and sizes | | | |
|---|---|---|---|---|
| | $M_1$ | $M_2$ | $M_3$ | $M_4$ |
| Height, mm | | | | |
| minimum | 603 | 708 | 845 | 1,033 |
| maximum | 1,000 | 1,210 | 1,470 | 1,845 |
| Extension range, mm | 397 | 502 | 625 | 812 |
| Thickness of seam, metres | 0.8-0.95 | 0.9-1.15 | 1-1.4 | 1.2-1.8 |
| Service load, tons | 35 | 35 | 35 | 35 |
| Yield rate, mm | 75 | 75 | 75 | 75 |
| Weight, kg | 29.5 | 32.6 | 38.4 | 48.5 |

A schematic diagram of the wedge arrangement in the expansion post is shown in Fig. 159. It consists of two clamps *1*, connected by strips *2*, two vertical wedges *3* and level wedge *4*.

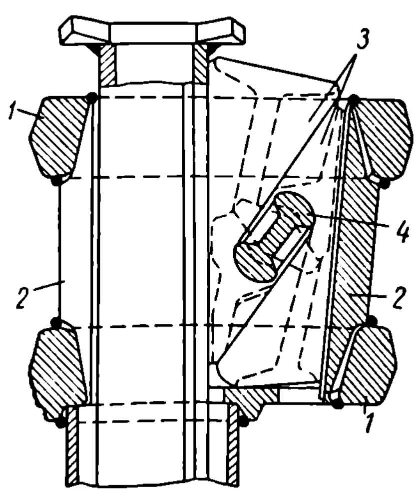



*Fig. 159.* Extension metal post wedge lock

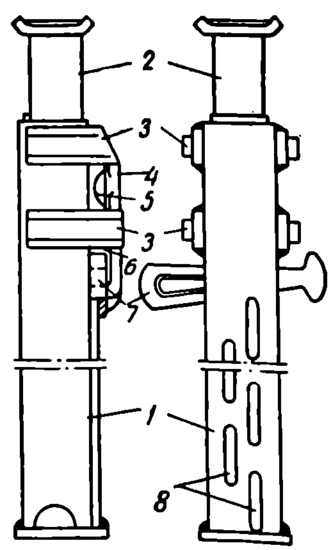



*Fig. 160.* ДонУГИ metal post

The post ДонУГИ(СДТ) also belongs in the category of yielding telescopic posts (Fig. 160). This is comprised of body *1* and extension *2*. The wedge lock consists of two clamps *3*, thrust strip *4*, working or thrust wedge *5*, gib *6* and level wedge *7*, all of them placed one on top of the other in the body of the post. In order first to tighten the post in place, wedges are driven into holes *8* on the opposite side of the body in the process of installing the post. ДонУГИ posts for low seams, 0.5-0.7 metre thick, are intended for a load of 25 tons and weigh 14 kg each. Posts of similar design are made for seams 0.7-2.3 metres thick.

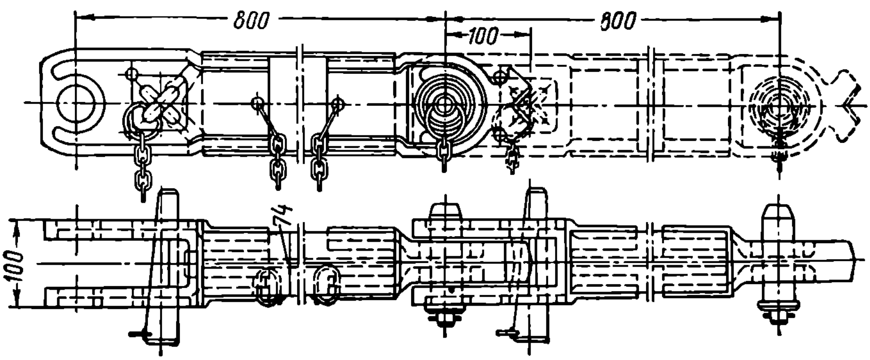



*Fig 161.* Swivel-type metal cap

Metal props are more economical than the timber, providing they are shifted from place to place not less than 40-50 times.

By the nature of their action the above-mentioned wedge-type posts should be included in the group of "rising resistance" designs. There may also be, however, posts of "constant resistance" type, with a high initial supporting capacity which changes but little after further extensions of the telescopic portion, and this is of great significance in roofs undergoing considerable subsidence.

In modern coal mines metal props are used very widely. In the Donets coal fields, for instance, their number at the close of 1954 exceeded 336,000, while in West-German mines there were about 1,300,000 such posts in use.

Timber *caps* of the sets at working faces can also be replaced by *metal*. To reduce the weight of their individual elements and to facilitate their transfer and installation they are made of a *split-swivel* type (Fig. 161). The weight of metal caps is 20-30 kg. This can be cut down considerably by making them of light alloys, with aluminium as a base element.

Use may also be made of sectional *metal* cribs (Fig. 162). Such cribs have two wedge beams with chamfered ends at which the wedges are held in place by catches.The wedge key is released by striking this catch with a hammer, the crib is freed from pressure and can subsequently be dismantled.

We have seen before that closely set rows of posts or "breaker props" are used to effect roof-caving control in walls with timber support. For the same purpose the State Institute for Designing Coal Equipment has suggested metal structures in the form of solid expanding props. At first, because of the purpose they served, these structures were called "metal prop walls"; now they are called simply *breaker props*. One of the most upto-date designs of this type—mechanised organ support MOK (Fig. 163)—includes a base and superstructure whose surfaces contact each other along an inclined plane. The superstructure is furnished with a screw extension device and a slab held against the roof. The base has a mechanism enabling to keep both parts of the prop in a definite position or to disengage them.

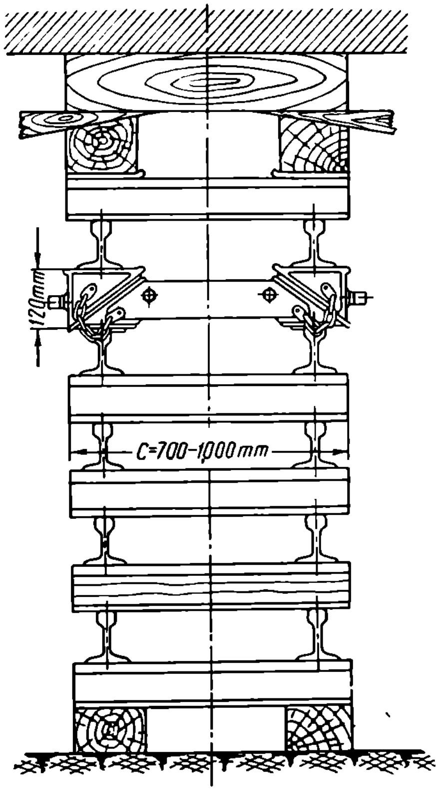


*Fig. 162.* Sectional metal crib

The last operation, that is, removal of pressure from the prop, is effected by pulling a rope line from a special prop shifter. By means of the same rope line the free prop is set in a new position, closer to the face.

Breaker props of this kind are manufactured in two types and sizes for seams from 1 to 1.8 metres thick. They are designed to bear a maximum load of 350 tons. The weight of the prop itself is 0.4 ton.

The *prop shifter* (Fig. 164) has a drum for pulling rope driven by an electric motor employed in coal-cutting machines. During the operation the shifter is reliably secured by its hydraulic anchor post.

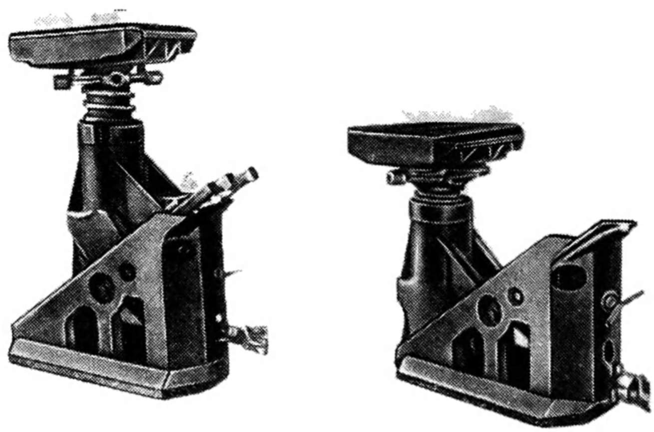


*Fig. 163.* MOK metal breaker props

*Supporting blocks* can be used instead of prop walls. A vertical section of one design is shown in Fig. 165. This block is made of mounting or base *1*, medium extensible section *2* and upper slab or plate *3*, which bears pressure coming from the roof. The base and the medium section are connected by a screw thread, this permitting to vary the height of the block from 10 to 40 cm, depending on the size of the block. The block is maintained at a preset height by wedge key *4*. There are five OKY supporting block models now being manufactured, their maximum height ranging from 600 up to 1,300 mm. The load-carrying capacity of such a supporting block is 150 tons. In a coal wall the blocks are set up in rows, like the props of organ timbering. Supporting blocks are shifted by hand with the aid of handles *5*. The use of blocks reduces consumption of supporting props, but these blocks are heavy (124-202 kg) and their handling and shifting presents many inconveniences.

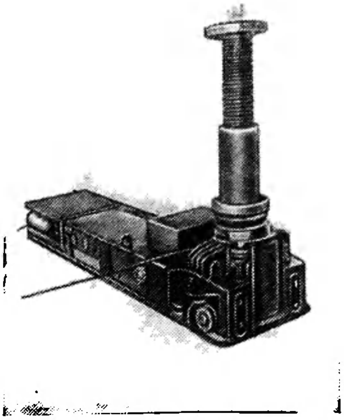


*Fig. 164.* Prop shifter

In describing organ timbering above, we stressed that it was relatively more rigid than the cribs. Metal break line props are naturally even more rigid, which is of special importance for roof-caving control. The use of metal cribs, props and supporting blocks in the mines of the Donets coal fields has already shown that they make roof-caving control feasible in conditions where organ timbering was of no avail. This is very important, for it suggests ways of considerably restricting the use of the labour-consuming method of roof control by pack walls built of rock drawn from lateral entries.

In discussing the use of the above-mentioned metal supports at production faces, it should be borne in mind that there may be instances when their use is impossible. If the seam contains thick intercalations, the mined-out area is encumbered with waste that hampers the shifting of metal support sets. When wall rocks are unfirm, the metal sets tend to sink deep into walls when subjected to rock pressure, and this also complicates the shifting of sets. The use of metal supports can also prove extremely difficult in sloping and, still more, in high-pitching seams.

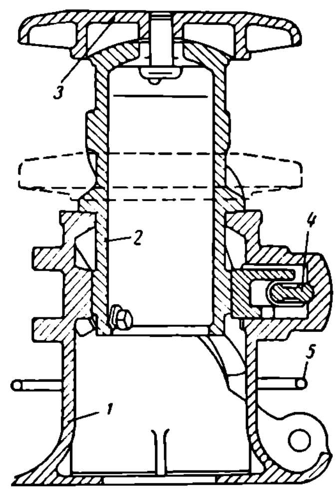


*Fig. 165.* Break-line set metal support block

Quite original and remarkable are the Soviet inventors' efforts to design *mechanised movable* supporting sets for continuous faces. Back in the 1930's A. Zhuravlyov suggested construction of a solid metal shield consisting of individual sections capable of protecting the active stope area from caving in and of being moved immediately after coal extraction. The inventor hoped that the pressure of caving rocks would be sufficient to make the shield move. The hope was not justified, hence the tendency to *mechanise* the shield. Most suitable for this purpose were hydraulic units, jacks, operating under the pressure of liquid (oil) supplied by pumps of a special design at tremendous pressure—several hundred atmospheres. Such a mechanised shield, designed by Zhuravlyov and Pokrovsky, was some years ago tested in the mines of the Karaganda coal fields, but the tests did not lead to its mass utilisation.

Of considerable interest are field tests in the Moscow coal fields of a shield invented by engineers Ziglin and Geller. This is a set designed to support a narrow strip of roof in an active stope area and to protect the latter from caving in or from subsiding rocks on the side presenting the greatest hazards.

The shield consists of individual link-connected sections. They are arranged in a single row along the coal wall and moved simultaneously,

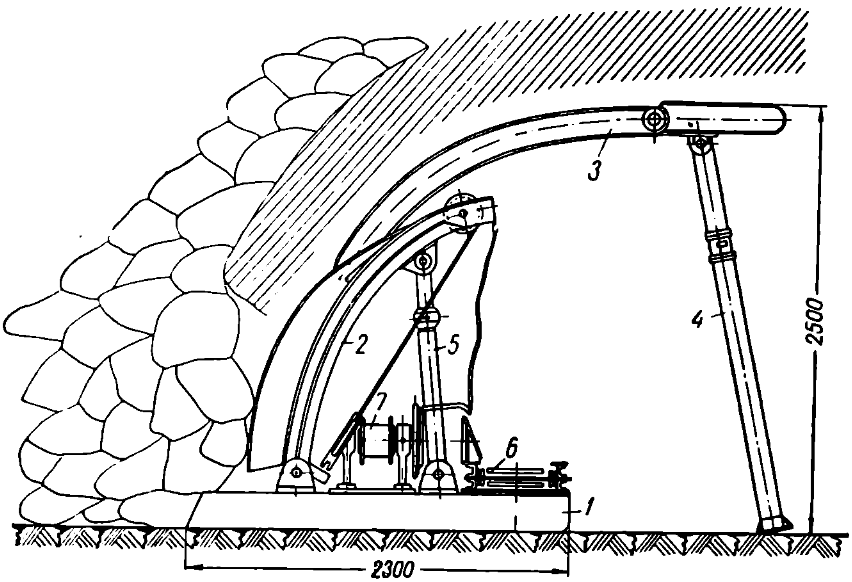



*Fig. 166.* Щ-52 movable mechanised shield

after the extraction of the entire coal bench, with the aid of ropes and winches placed in the entries.

Three winches are needed to move a shield 50 metres.

Inventors have created several designs of such a shield. Fig. 166 shows model Щ-52. Each of its sections consists of bed *1*, body *2* and two sliding deflectors or visors *3*, supported by posts *4*. Uprights *5* support the body of the shield. The bed accommodates flight conveyer *6* and hoist *7* moving the deflectors.

Coal is mined by blasting and about 40 per cent of it is loaded onto the conveyer automatically. When coal from the face has been drawn, the visors are slid into a position indicated in Fig. 166 with the aid of rope lines and the hoist. The shield is then moved into its new position. During this operation the front ends of the visors remain stable, while articulated sections *3* are let down and slide along the body of the shield. The pace or space interval of the shield transfer is one metre. One major advantage of the shield method of support is that it completely eliminates timber consumption in coal walls.

Elaboration of the final construction details of the above-mentioned shield and the methods of its field operation are still going through a stage of improvement. Thus, for example, there are tests of devices providing for the extension of visors not with the aid of ropes pulled from a hoist but with that of a hydraulic mechanism.

The main idea underlying the design of this shield is not to make it support the roof but merely protect the active stope area from roof rocks caving in behind the shield (*protective timbering*). But there have also been suggestions for building a movable mechanised support capable of bearing pressure coming from the roof at the coal face (*supporting timbering*).

This type of support includes the one known as МПК (mechanised movable support), proposed by Abroskin, Bondarev and Dashevsky for operation in coal walls of gently sloping seams from 1 to 1.7 metres thick when they are worked by combines.

The principal features of the МПК support are as follows (Fig. 167).

The support comprises separate disconnected sections *3*. Each is made of a tubular steel column with a solid cantiliver head piece 2.3 metres long and 0.45 metre wide welded to the upper end. The head piece overhangs the coal face. The upper and lower parts of the column are connected by a female screw thread. To alter the height of the column, it suffices to turn the lower part, which is shaped like a spherical shoe. The actual turning is effected with the aid of a small crowbar inserted into the holes provided in the shoe. In addition to this the column is furnished with a wedge key to provide for an outward thrust. The sections are moved to a new position at the breast of the coal face by special hoist ("shifter") *4*, for each of them weighs about 0.5 ton. The sections are moved forward parallel to the progress of the coal combine, approximately 15-20 metres behind it. The transfer of one section takes about three minutes. A stand-by shifter *5*, is available in the wall.

Coal is brought out from the face by "winding" conveyer *2* of КС-1 type, which can be moved to the face with the aid of the above-cited shifter without its being dismantled (see Fig. 167). The roof over the mined-out area is subject to caving immediately after the support, whose bearing capacity per metre of the wall reaches 330 tons, has been moved to a new position. The active stope area behind the mine combine is temporarily supported by light extensible posts. The use of the МПК support eliminates consumption of mine timber almost completely.

In the same category is the mechanised movable support invented by V. Vorobyov, T. Gorbachov, I. Patrushev and F. Kufarev, which has lately been put to the test in the mines of the Kuznetsk basin. Inasmuch as in this design the movable powered support is combined with mechanised extraction of coal with the aid of a coal plough, the inventors gave the whole arrangement the name of Kuzbas Coal Combine.

This combine is made of the following assembly units (Fig. 168): coal plough *1*, driven by hoists *2*, self-propelling mine support *3*, conveyer *4*, control board *6* and hydraulic pump plant *7*. All these

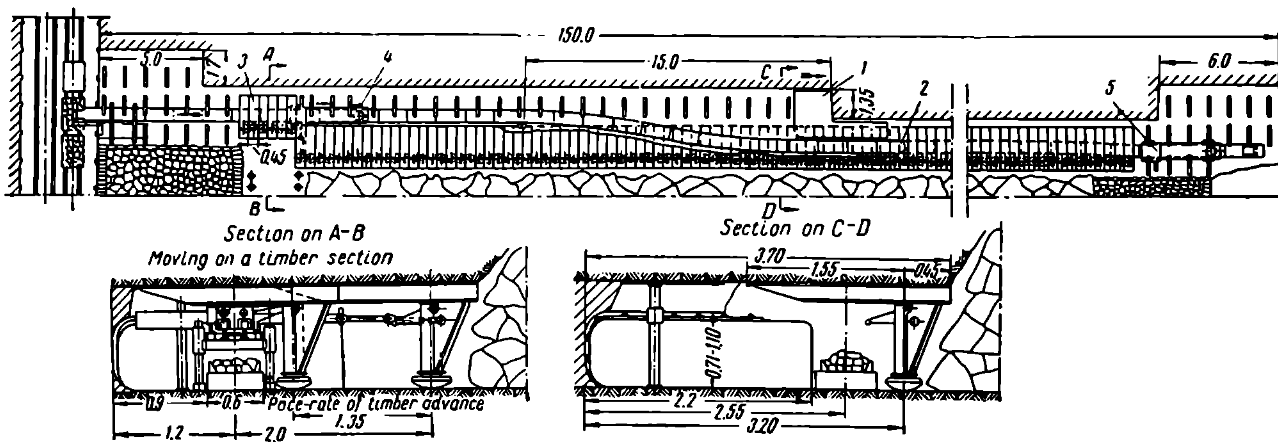



*Fig. 167.* Powered (mechanised) movable support

1—Donbas coal combine; 2—KC-1 conveyer; 3—МПК support section; 4—shifter; 5—stand-by shifter units go to make a machine operated by one man from a central station.

Moving along a coal wall, the plough cuts a strip of coal 20 cm thick. The face conveyer and overloader transfer coal to gate conveyer

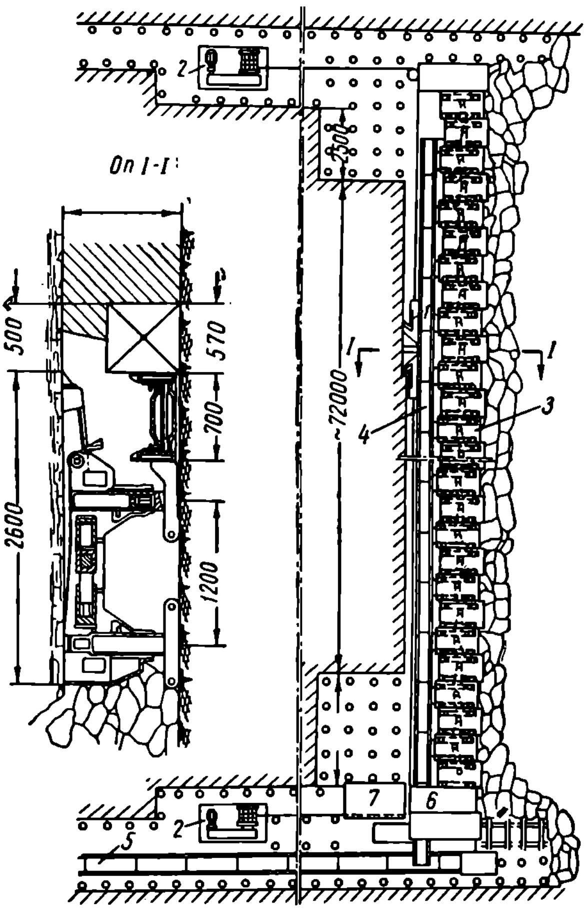



*Fig. 168.* Different positions of the Kuzbas coal combine in the wall

*5* installed along a strike entry. When a coal strip has been cut from the breast of the face, the combine moves forward, simultaneously shearing the coal left in the roof after the passage of the plough.

The roof of the face is held in place by the sections of the support. Thus, as proposed by its inventors, the combine is meant to extract, load and transport coal from the face, and support and control the roof in one process based on the principle of a continuous operation.

## 4. Filling of Mined-Out Areas

Worked-out space in walls with continuous faces (longwalls) in slightly inclined and sloping seams is filled primarily with materials sorted out in the stope itself.

A method widely used to obtain packing is that of driving *lateral entries* or *dummy roadways* (Fig. 169). It implies pushing forward rock entries *1,1* with brushing in a mined-out space, the sole purpose being to obtain waste for filling. They require but light timbering and are not supported in the open goaf. Their roof at the face can be supported by individual props (relief posts).

Roof brushing in such entries is more convenient than floor brushing, but this disturbs the continuity of rocks in the back of the entry and is attended by increased rock pressure at the production

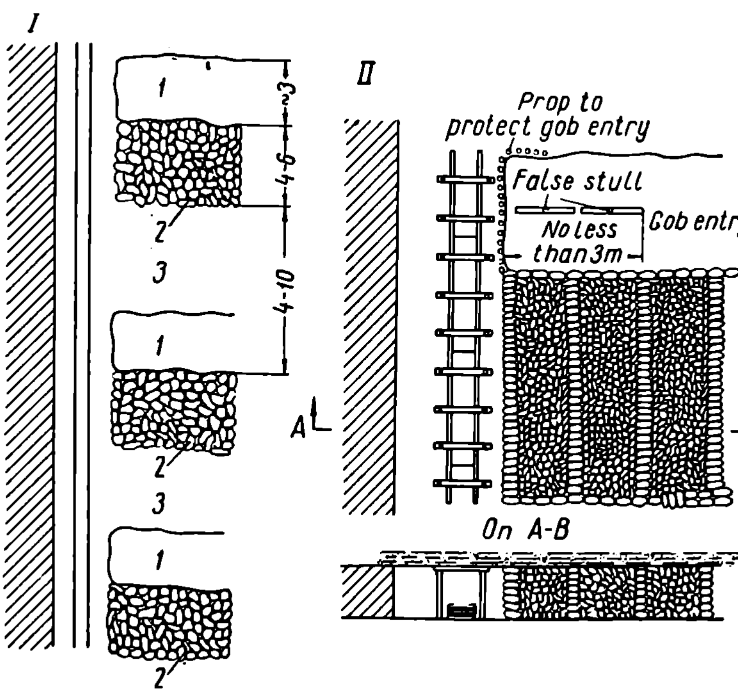



*Fig 169.* **Lateral entries and pack walls**
*I*—general view; *II*—details

face. But it is the back that is ordinarily ripped in this entry. The ripped rocks are arranged on both sides of entries in the shape of *pack walls* *2,2*. To lend stability to them, the packing strips are built on the sides of large blocks of rock in the shape of walls (Fig. 169, *II*), between which smaller lumps of rock are placed less carefully but so that the fill should reach the back and the pack wall be made a reliable support for the roof. The width of pack walls depends on the firmness of the roof and is usually assumed to be about four times as thick as the seam.

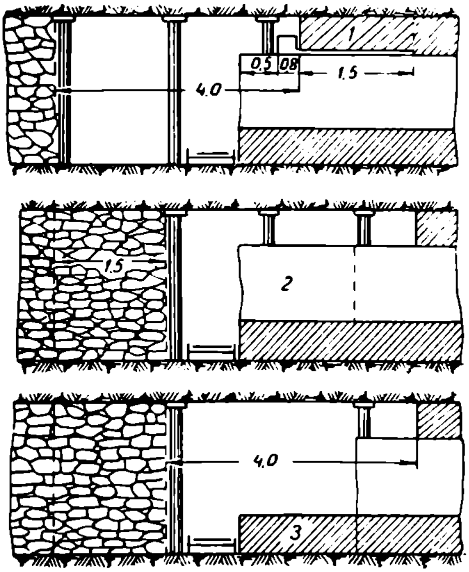



*Fig 170.* Complete filling in mining a seam with gangue intercalations

When the pack walls of neighbouring entries meet, the mined-out space is said to be completely filled. To cut down the cost of driving lateral entries, they are more often than not spaced so as to leave unpacked areas *3,3* between them. In other words, the filling is incomplete.

Although the extraction of waste filling from lateral entries is practised quite widely in the mining of low seams, this method of roof control nevertheless has one major disadvantage—it is highly labour-consuming because the packing is done by hand.

The attempts to mechanise this extremely labour-consuming operation with the aid of scrapers have been unsuccessful. Recently V. Fishchuk and P. Tereshchenko elaborated special machines for the diagonal stowing of the waste obtained in ripping an airway driven right after the coal wall. Successful field tests of these "mechanical packers" will bring the task of mechanising pack wall stowing closer to its accomplishment. Pack walls built up of brushed rocks are also frequently used to protect mine workings.

When seams with considerable intercalations of barren rocks are mined, the amount of waste available at the face may be sufficient for a full fill. One example is the mining of the Begly anthracite bed at one of the pits of the Rostov coal field. The bed of a total thickness of 1.7-2 metres is divided into two bands by a gang intercalation one metre thick (Fig. 170). Upper band or bench *1* is undercut first and holes in coal and in gangue interlayer *2* are drilled. Broken coal is loaded onto a conveyer, when mining proceeds up the dip. Along with the extraction of coal charges are shot in holes drilled in the gangue interlayer, and loosened rocks are thrown over the conveyer and stowed in the worked-out area. Then the bottom coal bench *3* is blasted without any preliminary undercutting. The mode of face support is seen clearly in the drawing. Because of the considerable thickness of the intercalation the volume of waste obtained during the breakage is sufficient to secure a full packing capable of supporting the back of the seam.

## 5. Artificial Caving of the Roof

We have already seen that the roof control can be effected by its *artificial caving*. We shall now discuss the various methods employed to achieve this end.

In order to induce roof caving, the props placed behind the special support have to be completely removed from the mined-out space whenever possible. After that the rocks over the corresponding area start caving under their own weight. But since this special timbering is much stronger than that made of individual posts and can fully support the rocks overlying it, caving causes a rupture (*break*) in the subsiding mass of rocks along a certain plane. As soon as such rupture has occurred, cohesion between the mass of rocks settling over the mined-out area behind the special timbering and those lying closer to the face becomes disturbed, and that is why when they start subsiding and caving they do not entrain the others. Due to this rock pressure over the active face grows weaker.

By regularly repeating the operation above, that is, by setting up new special timbering at definite intervals as the production face advances and causing caving by pulling out the props it becomes possible systematically to maintain rock pressure over the active face within permissible limits and thereby eliminate the encumberment of working faces with caved-in rocks. This, in fact, constitutes the main principle underlying *roof control* by artificial caving.

The geological structure of rocks overcapping the worked coal seam and their properties make caving now easier and now more difficult. Clay and sandy shales of medium firmness lend themselves quite well to roof control. Hard shales and sandstones cave in much more readily when they are broken up by jointings.

The *space interval* or *rate of caving* is determined in each individual case by experience. It ranges widely from 1.5 to 10 metres, according to local conditions, and that corresponds to one or several machine cuts.

*Artificial roof* or *back caving* is not altogether a safe operation and it is, therefore, entrusted to experienced men only. Caving itself should be performed under the direct guidance of a person belonging to the supervisory technical staff of a rank at least equal to assistant mine-section superintendent. It is he who sanctions other operations in a coal wall whose angle of pitch does not exceed 18° and who allows workers to be at a distance not less than 30 metres from the section marked for caving.

*The pulling out of props*, an operation usually requiring three or four workers, as a rule begins in the rear upper corner of the area where it is performed. When props are pulled out in a seam pitching at more than 15°, the job is effected up the dip so as to prevent falling and sliding rock blocks from knocking timbering down. Timber is knocked down in the diagonal direction so that the workers doing it stay at all times in the supported portion of the mined-out area. Before beginning, the timber pullers carefully examine each post and pull out only those which can be removed safely. To preclude rock cavings, the posts serving as the chief support of the blocks of rock and standing near the place where rocks are fractured are not removed. Besides these posts are others called safety or *signalling* which are not pulled out and the purpose of which is to warn the timber pullers of danger. As the ground starts to move, it causes the "signalling" posts to crackle and thus enables the workers to find shelter in good time. The posts are knocked down by sledge-hammer blows at their upper or lower ends or, if they are tightly jammed, with an axe. The pulled-out posts are brought up to a new line of special timbering.

Before the posts are pulled out, the old cribs are shifted to a new position, while the organ supports near the old "rib" of the break line are either not removed at all, because this operation is dangerous, or else are drawn off only partially.

The transfer of cribs and the pulling out of props mean a considerable economy of mine timber, since this makes it possible to preserve from one-quarter to half and even more of the total number of cribs and posts. A man working in a seam 1-1.5 metres thick pulls out from 80 to 140 posts in a shift.

Despite the important advantages gained by pulling out the posts—saving of mine timber and adequate roof caving control—it is by far not always that this operation is undertaken, since this requires a set of definite conditions, namely:

1) the roof should not be subject to sudden, spontaneous collapse, its caving should be preceded by definite signals, such as preliminary crackling of posts after the rocks have started sagging and bumping sounds following the disintegration of solid rocks;

2) the goaf should not be packed with any considerable amount of the mine-fill that would hamper pulling out the posts;

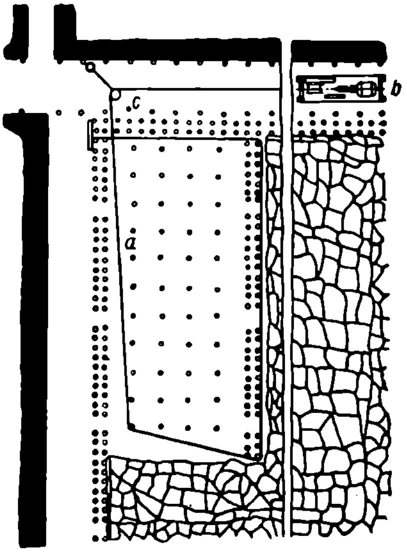



*Fig. 171* Mechanised post pulling

3) the pulling out of posts is possible only in slightly inclined seams, when the falling blocks of rocks remain on the spot and do not slide down as is the case in sloping and high-pitching beds, where the drawing off of posts is not practised;

4) operations involving artificial caving are difficult to perform in extensive walls, since delays are quite probable in stoping in the process of induced roof-caving and possible collapse of the back;

5) the bed should not be too thin (not less than 0.7 metre), since in a thin seam the movements of workers engaged in pulling out of posts are very much restricted.

But in thin seams the back quite often does not cave at all, for when the roof starts sagging over a worked-out area it has, because of the insignificant height of this space, enough time gradually to settle down to the bottom before any fractures and ruptures actually occur in it.

A factor which sometimes facilitates the gradual uneventful filling of the mined-out space is the heaving of the bottom.

If, for some reason, the posts are not pulled out or the roof is stable and does not cave in even after the cribs have been transferred and the posts pulled out, to avoid subsequent mass caving the overhead rocks are sometimes caved by means of explosives, charged in several holes drilled behind the cribbing.

Attempts have been made to *mechanise the knocking down* of mine timber (roof caving by *machines*) (Figs 171 and 172).

Mechanised knockdown of mine timber consists in demolishing supports in a worked-out area either by tipping or breaking posts with the aid of steel cable *a*. One end of the cablo is secured while the other is gradually wound around the drum of hoist *b* set up in an airway. The cable is guided by idle roller *c*. It is not only the face sets, but breaker posts too that are thus removed.

Mine timber pulling hoists or tuggers (Fig. 172) manufactured by the Kuznetsk Machine-Building Works have a drum with a diameter of 400 mm, are 3 metres long, 0.9 metre wide and one metre high. Their rated rope pull is 14 tons. Cable speed rate is 0.14 m/sec, with motor capacity of 20.5 kw. The breaking force for the cable used is 23-28 tons, its diameter—20-22 mm, it is of cross lay, made of thin wires (1-1.4 mm) with tensile strength of 140-160 kg/mm².

Artificial caving operations, that is, fixing and handling of the cable, putting up idle rollers, tugger control, etc., are performed by a team of three or four men. Artificial roof caving proper in a coal wall 100 metres long lasts 20-25 minutes. The size of the caved area on strike should conform to the space interval of caving determined by experience—usually three or four cuts, that is, 5-7 metres.

The advantages of mechanical artificial caving are considerable. They are:

1) greater safety of operation;

2) complete demolition of timbering, ensuring full and rapid settling of the ground, and this, in turn, favourably affects the condition of the roof over the active stope area;

3) caving takes very little time.

This mechanised method of artificial caving, however, causes complete loss of mine timber. But practice shows that it is often possible to "thin" timbering by knocking down some of the posts (25-35 per cent) by hand before proceeding with mechanised roof caving.

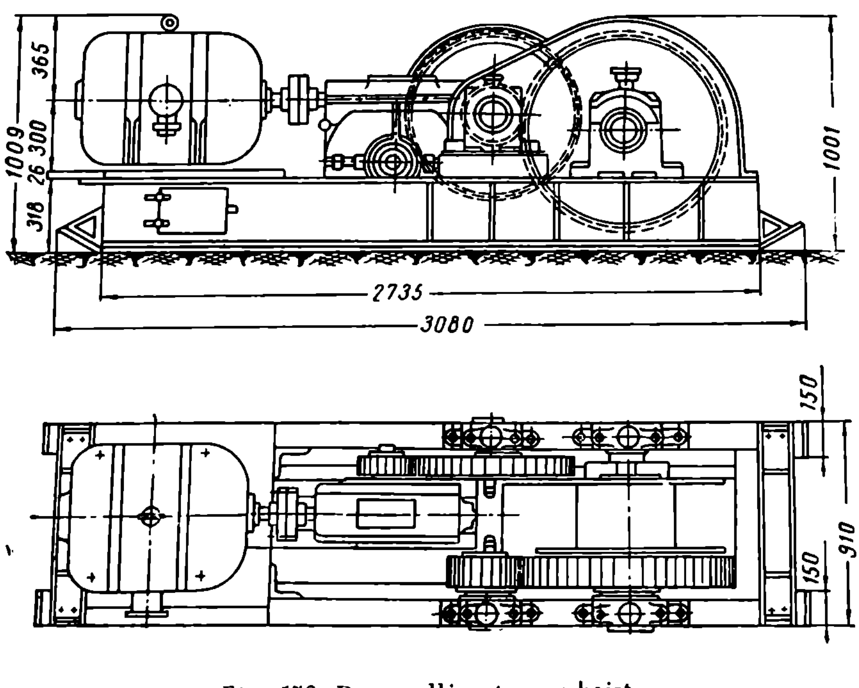



*Fig. 172.* Prop-pulling tugger hoist

Successful experiments have been carried out in recent years in the Moscow coal fields in drawing off mine timber with the aid of cables and electric tuggers.

### 6. Collapses of Working Faces and Retimbering

Normal, well organised work at the coal face may be completely disrupted in the event of a *collapse*. This happens when the condition of the occurrence of rocks suddenly deteriorates, for example, when faces unexpectedly approach a geologically disturbed section. Unfortunately, such collapses or goafs are due mostly to negligent attitude towards and insufficient care of the roof and timbering at the active face.

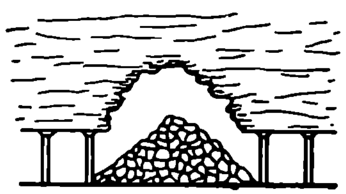


*Fig. 173.* Local collapse in a working face

In instances of minor local collapses (Fig. 173), cleaning up operations require careful removal of caved-in lumps of rock and installation of a new ordinary timbering, which is often reinforced by cribs. At any rate, it is imperative to prevent the collapse from spreading and this is done by setting up additional supports over the whole area of the face.

But sometimes it happens that collapses occur in such a fashion that great masses of rock start caving in at the very face (Fig. 174, *I*) and special methods must be employed for its retimbering.

To do this lumps of rock are cleared away with caution to an extent making it possible to set up inclined or battered props (Fig. 174, *II*). Under protection of this support the operation at the face is restarted and, as soon as possible, ordinary timbering (Fig. 174, *III*), followed by a breaker row or cribs, is put up. The collapses, however, may be so extensive that their clearing proves a rather difficult and dangerous affair, for, as the result of the collapse, fissuring of the rock masses occurs over the coal seam itself, near the very face. In such cases the site of collapse is not retimbered, but by-passed and to do so a new break-through (Fig. 175) is driven along the face at a distance of 1-1.5 metres from the site of collapse, wherefrom stoping operations are being started in the usual manner.

In individual instances valuable equipment (coal-cutters, combines, conveyer drives) may be buried at the production face following a collapse of the coal wall. This can be saved by pushing *cross-headings* from the above-cited by-pass break-through towards the sites where it was originally operating.

Those responsible for the conduct of work at the production face and in the mine should take beforetimes all the necessary measures to prevent and preclude collapses of coal walls.

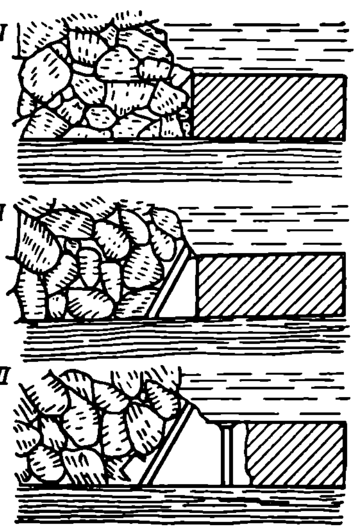



*Fig. 174.* Retimbering a collapsed working face

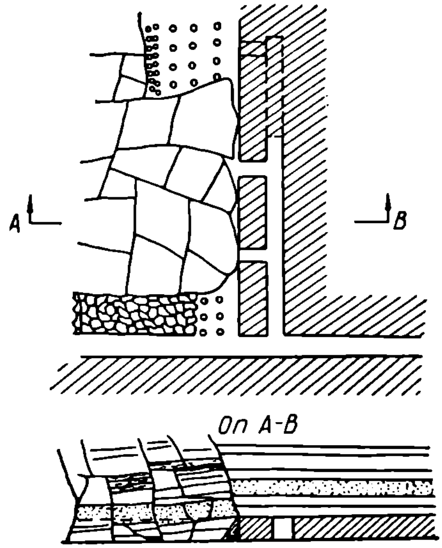



*Fig. 175.* By-passing a collapsed working face

## 7. Conveying Coal in Production Faces

In the early period of all-round mechanisation in the continuous production faces of slightly inclined seams it was electrically driven *shaking conveyers* that were used almost exclusively for coal transportation. The old chain-and-flight conveyers were considered too heavy, cumbersome and inconvenient to be moved about frequently in active faces. But the progress of the Soviet mine machine-building industry has in recent years given our mines some new types of chain-and-flight conveyers that are more suitable for the operation in working faces.

Up-to-date chain-and-flight conveyers have the following advantages over the shaking conveyers: they carry coal not only down a slope, but also in level sections and upgrade; their efficiency is higher. At present there are reversible-type chain-and-flight conveyers available, capable of delivering materials (coal, mine timber) in two directions. They work more smoothly and are less noisy because there are no reciprocating motions and constantly varying accelerations and decelerations, a feature common to shaking conveyers.

The CKP-11 conveyer (chain-and-flight conveyer of the reversible type, powered by an electric motor of 11 kw) (Fig. 176) has been disigned by the State Institute for Designing Coal Equipment (Chief designer—N. Samoilyuk). Its coal-carrying capacity is 60 tons per hour. It is 70-100 metres long, the speed of the drag chain is 0.4 m/sec, rated motor capacity—11.4 kw. The conveyer trough is 150 mm high, 500 mm wide at the top and 350 mm at the bottom. It is pulled by a stamped drag chain. The driving tumbler or head pulley is set up at the discharge station of the haulage entry, but if the pitch is in excess of 10°, it is better to put it in the upper section of the coal wall. The minimum thickness permitting the use of this conveyer in working seams is 0.65 metre.

More powerful chain-and-flight conveyers—the CKP-15, CKP-17 and CKP-20—have lately been put into operation to cope with the growing amounts of coal mined at faces worked by combines. The CKP-20's capacity is 100 tons of coal per hour and it is 150-170 metres long.

To facilitate loading, the conveyer board on the side of the mined-out area is higher than that on the side of the working face.

For operation in low seams the industry has organised serial production of $CKT_2=6$м and $CKT_3=6$ chain-and-flight conveyers. They are 100 metres long and capable of handling 40 tons of coal per hour.

The increasing employment of coal combines enhances the need for face conveyers whose design facilitates their shifting in a coal wall and takes into account combine operations and methods of roof control. Designing organisations are actively engaged in elaborating

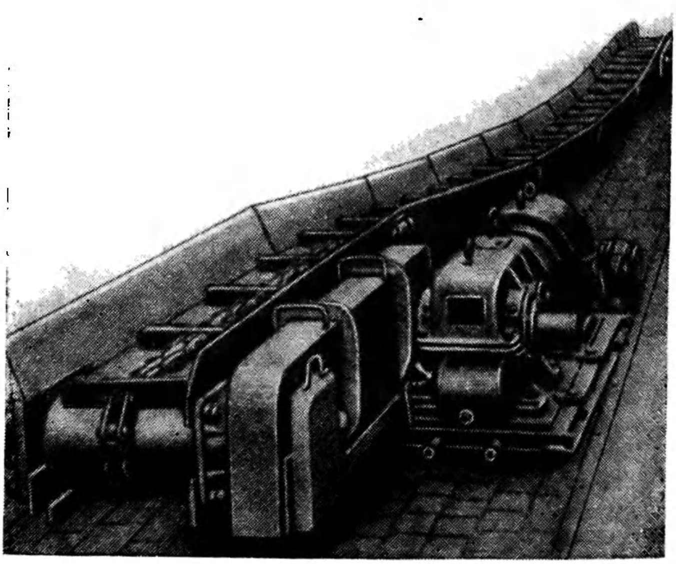


*Fig. 176.* Reversible chain-and-flight conveyer

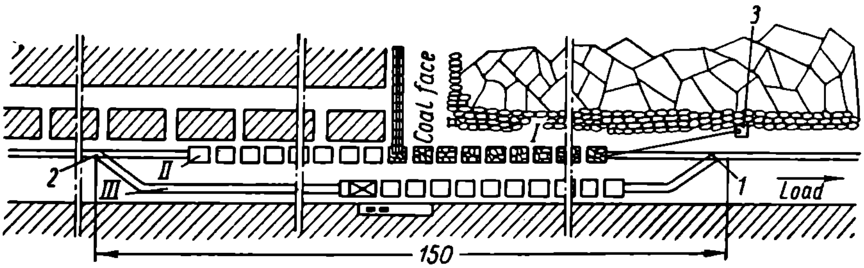



*Fig. 177.* Loading of coal from a face conveyer

and testing such conveyers. For example, there is the KC conveyer designed to operate with the Gornyak combine. It is 170 metres long, has a 20-kw electric motor and is capable of handling 80 tons of coal per hour.

*Belt conveyers* are not widely used in production faces because it is difficult to shift them and the rubber belt wears out quickly.

The loading of coal drawn from walls into mine cars is simple, especially when there are no entry pillars or when they are extracted simultaneously with coal recovered at the working face (Fig. 177) and coal is loaded into mine cars directly from a face conveyer. Pass-by tracks (100-150 metres long, depending on the length of trains) should be laid at the loading site. The empties, brought up for loading through entrance switch *1* with an electric locomotive at its head, move along run-around track *III* beyond switch *2* and are then brought to the empties section *II* of the loading track. After it is uncoupled from the train, the electric locomotive passes through switches *2* and *1* to the loaded portion of the track *I* and pulls the coal train. A low-speed trip-spotting hoist *3* (with rope speed of 0.15-0.3 m/sec) is employed for spotting cars during loading. Both this hoist and the conveyer in the coal wall are remote-controlled from board *4*. The work of the loading station and conveyer in a coal wall is thus supervised and regulated by one operator.

If there are sill coal pillars over the haulageway (Fig. 178), two short intermediate conveyers are set up near the loading station. In Fig. 178 digits *1*, *2*, *3*, *4* are track switches and *5* is the trip-spotting hoist.

*Scraper mucking* is unsuitable in the conditions existing in longwalls, since the per-hour efficiency of coal transportation drops very rapidly as the distance increases. At each given moment the scraper removes the material at one point only. By "tearing" and "stripping" the bottom of the bed, the scraper tends to increase the ash content in the coal mined. The scraper path is broader than permissible for conveyer transportation, which is not always possible because of rock pressure in the back. For these reasons no scrapers are employed at present in long coal walls. When conveyers are used for coal transportation at working faces, it is extremely important for timber sets to be arranged in straight lines.

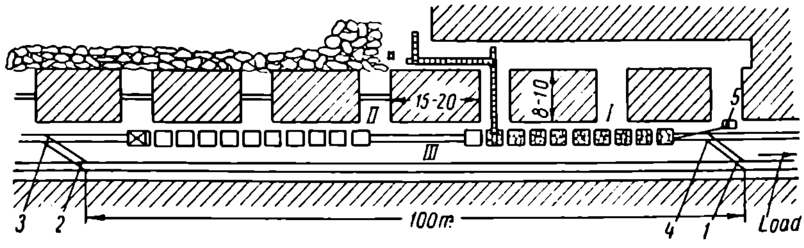



*Fig. 178.* Loading of coal from intermediate conveyers

Dumping of coal by gravity in *stationary steel trays* is possible with angles of pitch of not less than 20-25°, and its sliding along the bottom of the seam with an angle not less than 35°.

## 8. Ventilation and Lighting at Working Faces

*Ventilation* of a continuous rectilinear production wall is a simple matter. In a gassy mine, with the seam pitching at an angle of more than 5°, the air current should aerate the face by *ascending*, that is, moving from the lower to the upper entry of the given wall. This is due to the fact that the natural draught makes the ventilating currents flow in that direction, for, enriched by methane, air in production places becomes lighter. Therefore, with the air currents in the wall *ascending*, the natural draught will always cause air movement, even if the fans are switched off.

According to Mine Safety Regulations the velocity of air in a production wall should not exceed 4 m/sec.

Adequate *lighting* in a coal face is a necessary prerequisite for safe and highly efficient work.

In working places preference is given to individual *portable* electric battery lamps, which can be of two types—hand and cap ones. Hand lamps (Fig. 179) are used in coal faces of gently inclined low seams. Their illuminating power is 3 watts, weight—4.3 kg and burning time—10 hours.

Electric battery lamps of cap type are employed (Fig. 180) in working steeply pitching seams and thick beds. They weigh 3.7 kg and burn for 10 hours. Thanks to the mirror concentrating the luminous flux these lamps have an illuminating power of 20-30 watts.

In working-face conditions, *stationary* lamps (Fig. 181) have the

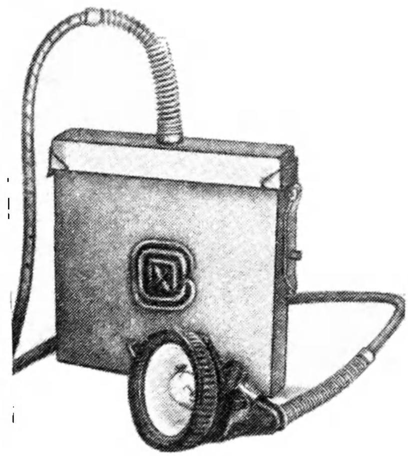


*Fig. 179.* Electric battery lamp

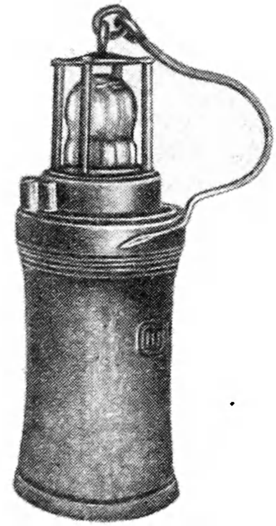


*Fig. 180.* Electric battery cap lamp

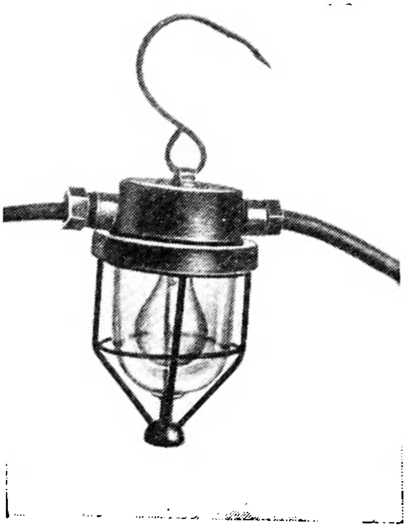


*Fig. 181.* Stationary luminaire for mine lighting

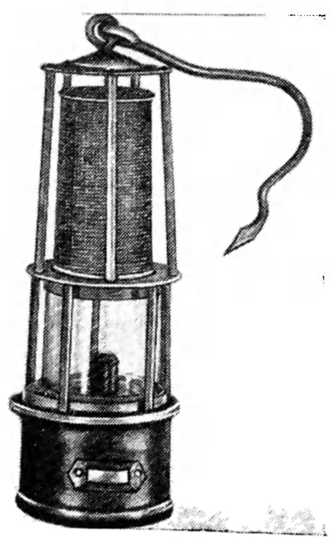


*Fig. 182.* Mine flame safety lamp

disadvantage of requiring frequent (almost daily) transfer of the lighting mains. Moreover, timber props and other objects throw immobile shadows that make light nonuniform. They are also inconvenient in blasting operations. The lighting mains can be transferred much more easily in short coal walls.

Electric lamps cannot be used to measure the firedamp content in the atmosphere of a mine. Therefore, when electric light is supplied by both portable and stationary lamps, there should always be one or

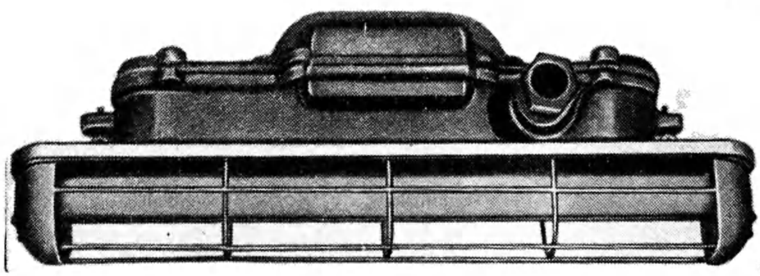


*Fig. 183.* Explosion-proof daylight lamp

two flame (benzene) safety lamps (Fig. 182), with which the methane level can be determined at any given moment. In electrically lighted working places these safety lamps are also needed to provide emergency lighting in case of current supply failures.

*Luminescent* lamps have lately been used widely for illuminating mine workings. They are used in haulageways, engine rooms, dispatcher and shaft stations, etc.

Fig. 183 shows a luminescent explosion-proof lamp.

CYCLIC STOPING

## 9. Definition of Continuous Cycle Operation

Production faces and stopes of development workings are the principal places of mining in active pits.

Fulfilment and overfulfilment of production programmes primarily depend on the successful operation of coal walls and timely and constant availability of reserves ready to be mined by means of driving development openings.

All the other numerous jobs and operations performed at a mine—transport and haulage of every kind, ventilation, drainage, power supply, etc.—have but one sole purpose: to ensure uninterrupted extraction of coal in walls, its delivery to the surface, and the driving and maintenance of development openings.

Proper organisation of work is of prime importance for normal operation of coal faces and development stopes. This can be achieved only on the basis of *cyclic operations.*

Individual operations in production faces recur periodically and in a definite sequence: after the undercutting, breaking and drawing of coal comes a new cut, then loosening and extraction of the mineral, etc. All other concomitant work in the wall, that is, timbering, roof control, shifting or transfer of equipment and machines utilised in extracting coal and its transportation from the coal face also recur in a definitely set order.

In other words, all these operations in a production face can be grouped in categories of successively recurring operations, that is, *cycles.*

Hence, a cycle in the production face of a coal mine can generally be characterised as a *complete course of processes and operations, performed in a definite sequence and necessary to mine coal over the entire coal face, the distance of its advance being provided for by the planned technical schedule.*

This general conception of cycle needs a more precise definition for the following typical cases:

For coal walls worked by mine combines or coal-cutting machines the cycle is an aggregate of operations effected between two consecutive undercuttings, or the cutting and loading of coal by a combine. When coal is broken by pneumatic hammers, the cycle is better defined as the sum total of operations performed at the face during its advance over a distance of two face timbering set shifts.

*The cyclic schedule* means *mining operations performed strictly and systematically in accordance with work cycles.*

Cyclic organisation of mining is simplest and most convenient when the sequence of operations in the cycle conforms to work shifts and *the whole cycle is accomplished within 24 hours* (the so-called *24-hour operative cycle*). In the case of the 24-hour cycle schedule each miner, at least for a certain period of time, works every day in the same shift, which is very important for keeping up regular and rhythmic operations.

When coal is extracted from a wall in two shifts and the third is devoted to repairs and preparatory work, the introduction of the one-cycle schedule in the face is conducive to adequate exploitation of equipment and proper maintenance of mine workings and haulage tracks.

In coal walls where, for geological, technical and other reasons, it is possible to extract the planned daily tonnage of coal *in one shift,* the remaining two are taken up by repairs and preparatory work. This method of work has already been adopted at a number of mines in the Donets and Kuznetsk coal fields inasmuch as it reduces the number of auxiliary workers and facilitates still more running repairs and inspection of machines.

If local conditions permit obtaining the daily planned tonnage in one shift and another is sufficient for repairing and shifting equipment, it is possible, as suggested by engineer Kosenko, to alternate production and repair shifts with the idle one in-between. This will make it possible to perform 45 cycles per month (see Section 13, below).

## 10. Importance of Cyclic Operations

It is of extreme importance strictly to adhere to the cyclic operation schedule. This is necessary to ensure the regimen of a coal wall, section and the mine as a whole, the fulfilment and overfulfilment of coal production plans, improvement of efficiency and reduction of the unit-cost of coal mined.

When, for instance, the number of cycles performed per month is less than planned, the advance rate of the working face also decreases, that is, actual coal output will be less than planned. And vice versa, *the more cycles the greater coal output.*

Compilation of work schedules helps establish efficiency standards for all men engaged at the face, determine the actual capacity of mine equipment and, consequently, of coal-wall output per cycle and per month. These calculations should naturally be done on the basis of optimal ratings for men and machinery over long periods of time. Hence, keeping to established schedules of cyclic operations makes it possible to achieve high labour efficiency, optimal utilisation of mine equipment and greater coal output.

Cyclic work means systematic, coordinated mining operations in the coal wall. Under the cyclic schedule, they are not performed sporadically and as occasion offers itself, but at a definite time and place. The cyclic method does not allow for rush work. It requires the work place to be maintained in proper condition and the machines and other face equipment to be given adequate attention and care.

The cyclic method ensures prerequisites for highly efficient teamwork, since it essentially presupposes that each miner does the job assigned to him. It ensures timely preparation of the work place and maximal productivity of labour.

An important guarantee for the successful cyclic operation of a coal wall is thorough preparation and organisation of work in conformity with local conditions and due attention to requirements enumerated below.

The cyclic method lends rhythm to the course of mining operations. This contributes in a great measure to the safety of miners.

Accidents are most frequent when there are no order and regularity in work. In the case of cyclic organisation, mining operations recur uniformly and safety measures, therefore, become traditional and stable. This is the best guarantee against accidents.

## 11. Planned Work Diagrams and Labour Distribution Charts

Proper work in coal faces, based on the cyclic method, should be carefully planned out both technically and organisationally.

A mechanised coal face, particularly if it is long, is worked by many miners of different specialities. It is equipped with machines, mechanisms, electric and pneumatic tools, lighting mains and lamps. Constant care should be exercised to ensure adequate support at the face, supply of mine timber, roof control and ventilation. In addition to this, work places in coal walls, and this distinguishes mines from other industries, are not permanent but *mobile*, since they constantly advance.

For these reasons distribution, sequence and coordination of mining operations in the wall should follow a *work schedule*, elaborated in detail and often of a rather complex nature.

This schedule, determining the sequence of operations in a coal face, is more comprehensible if presented graphically, in the form of planned work diagrams and labour distribution charts. These two are usually known under the common name of *operation chart.*

The *planned work diagram*, showing distribution in time and space of operations performed at a coal face, is compiled in the following manner (Fig. 184):

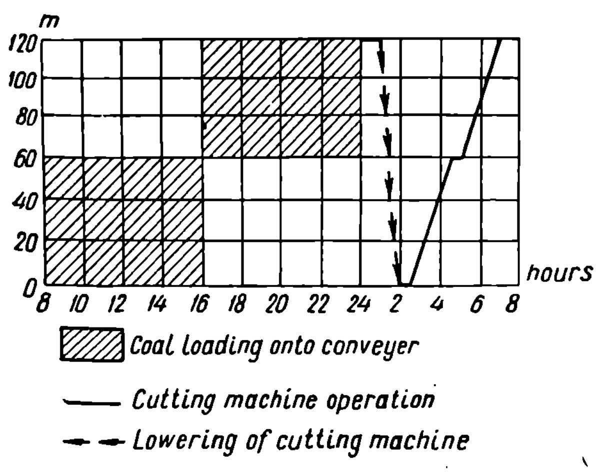



*Fig. 184.* Planned work diagram for a coal face

The duration of the cycle in hours is plotted on a definite scale on the horizontal axis. Consequently, with but one cycle a day it suffices to mark off the 24 hours. To facilitate readings, vertical thin lines are drawn through the points indicating hours. The day is split into three eight-hour shifts: I—morning shift, usually from 8 a.m. till 4 p.m.; II—day shift, from 4 p.m. till midnight, and III—night shift, from midnight till 8 a.m. It is best to start reading with the beginning of the first shift, that is, 8 (or 7) o'clock in the morning.

In the same drawing, the length of the wall is plotted on an arbitrary scale along the vertical axis on the left, with divisions every 10 or 20 metres, from which horizontal lines are drawn. In Fig. 184 the coal wall is 120 metres long.

To have a better idea of the way mining operations are depicted in a planned work diagram, let us, for example, follow the movements of a coal-cutting machine.

Let us assume that, in given working conditions, the face is undercut during the night shift and that at its beginning the coal cutter is in the upper portion of the wall. Its operator arrives at the face at midnight and spends an hour on checking and lubricating the machine, minor repairs, changing of teeth, etc. Therefore, if the position and operation of the cutter on a diagram are to be depicted by a straight line, it will be represented by its section on the axis—between midnight and 1 a.m. The descent of the machine starts at 1 a.m. On the diagram this operation should be shown by an inclined line (time goes and the machine moves along the wall).

If the descent of the coal cutter lasts an hour, the operation should be shown as indicated in Fig. 184—by arrows. Let us further assume that the initial cutting of the seam starts at 2.30 a.m. This operation is depicted by a continuous inclined line. But, then, let us suppose that, on account of local conditions, say, hard coal, the machine has to be stopped in the middle of the wall for half an hour for a change of teeth, lubrication, to cool the electric motor, etc. This stoppage will again be depicted on a diagram by a short horizontal section. The machine will then advance further and finish cutting the entire wall, say, by 7 a.m. The line of its descent is steeper than that of cutting because the machine descends at a higher speed.

At each given moment the cutting machine is engaged at some definite point of the wall. Hence, its operational schedule in the face can be represented by a *straight line*. The same also holds true with respect to cutter-loading machines.

Identical operations carried out simultaneously along the entire length of the face, or at least along a portion thereof, can be represented graphically on the diagram by *rectangular areas*. The height of such areas shows in what section of the long wall any given operation is effected simultaneously, while the length of the horizontal side shows the time interval in hours in which this operation is actually performed. One illustrative example is hand-loading onto a conveyer. Fig. 184 reveals that in the given case coal is loaded onto the conveyer in two shifts:—in the morning from the lower portion of the wall, in the day from its upper half. To make them more vivid such areas in the diagram should be hatched.

Straight lines and areas, like the ones described above, can be used clearly to represent all the other mining operations in a coal wall. For clarity's sake, individual processes are depicted in a variety of ways: by continuous and dash lines, crosses, etc. (Figs 185-187). These conventional signs should be as simple as possible.

Thus, a planned work diagram depicts the progress of all the basic mining operations performed in the wall in the course of a cycle. It vividly demonstrates the sequence and coordination (combination) of these processes in space and time.

To make the picture clearer, the face's position at the beginning of a shift is sketched alongside the diagram.

Another, very important type of graphic illustration of work done at the coal face, supplementing the planned work diagram, is the *labour distribution chart* (Figs 185-187). Time distribution for one cycle is shown in the chart as in the planned work diagram. The horizontal lines are used to list the workers by professions, their number in each shift and the whole cycle, and the duration of their work. The latter is depicted by thick horizontal lines. The labour distribution chart is best placed below the planned work diagram, on the same sheet.

The planned work diagram and labour distribution chart should be supplemented by *a table of technical and economic indices*, which are expected to be achieved in fulfilling cyclic operations on schedule, in conformity with planned organisation of work (see below). These figures are necessary for plans and estimates, as well as for the appraisal of the ultimate results of the work done in the coal face.

In addition to *projected* work diagrams, labour distribution charts and technical and economic results there are also *recorded* work diagrams and schedules, labour distribution charts and tables of technical and economic indices containing data on the fulfilment of the plan. The comparison of projected and recorded diagrams and charts makes it possible to supervise mining operations, take note of any possible shortcomings, expose their causes and take account of the ultimate results of the work done.

## 12. Compilation of Planned Work Diagrams, Labour Distribution Charts and Tables of Expected Technical and Economic Indices

In general, cyclic operation schedules are compiled as follows.

The amount of work to be done and coal to be extracted are established with due account to the geological and mining conditions prevailing in the given face. Then, comparing (see below) expected labour efficiency and the volume of diverse work, one proceeds to calculate the number of workers of different professions required for one operation cycle. After that these men and the mining equipment are distributed according to shifts, an effort being made to combine mining operations in a 24-hour cycle and to use the equipment as fully as possible. For clarity's sake, the outlined organisation of work is shown in the planned work diagram and labour distribution chart. Lastly, the expected technical and economic indices are calculated and tabulated.

Let us now turn to details.

1. The *geological conditions* to be considered in the cyclic organisation of work include: the angle of pitch of the seam; its thickness (aggregate and minable, in metres); the structure of the seam (relative position of coal benches and gangue intercalations); coal yield in tons per sq metre of the seam; hardness of coal; specific designation of wall rocks and characteristics of their stability; abundance of gas in the seam (gas category).

2. *Mining conditions*: the length of the wall (in metres); coal-face advance per cycle, that is, the width of the coal band broken in one cycle (in metres). The determination of this magnitude depends on the methods of coal breaking. With cutter-loaders (mine combines) it is their "cutting range", which is 15 cm less than the length of the bar. In the case of coal cutters, it is the effective cut depth which is approximately 15 cm less than the length of the bar when the latter is below 2 metres, and 20 cm less with longer bars.

The technical factors include the basic characteristics of mining machines and other items of mechanical and electrical equipment used in the coal wall. In other words, this requires knowledge of the rated capacity of mine combines, coal cutters, pneumatic hammers or electric augers, conveyers, coal ploughs, etc., their travel rate loaded and idle, overall dimensions and modes of propulsion; time required for lubrication, change of teeth and cooling of electric motors (this refers to coal-cutters and cutter-loaders). If drilling and blasting are resorted to at the face or in lateral entries (dummy roadways) to break and loosen coal and waste, this must be done in accordance with adequate technical specifications.

To determine the amount of timbering work to be done during one cycle and the volume of metal and timber to be supplied and consumed per cycle (minus that used again), there should be a technical specification for the support of any given face.

There should also be proper knowledge of the method of roof control by artificial caving, partial filling from lateral entries or dummy roadways, etc. The mode of special support (breaker props, metal or timber cribbing, etc.) should be made known too.

3. The data above are necessary to estimate *the volume of work* to be done per cycle.

Coal output per cycle is determined by multiplying the seam yield in tons per sq metre of surface by the length of the coal wall and its advance rate per cycle.

The amount of coal extracted from a wall in a month can be calculated by multiplying output per cycle by the number of workdays in a given month, which depends on whether the weekly operation at the face is continuous or not (see below).

To make sure of the fulfilment of the production programme, the planned output of coal should be somewhat below the rated one—by about 10-15 per cent. Hence, if the coal wall works at its best, the result is the accomplishment of one cycle per day and that will mean overfulfilment of the production programme.

The rated volume of work for all the operation items enumerated above is then determined.

4. Comparison of rated work volume per cycle and the men's actual efficiency per shift makes it possible to estimate the required number of man-shifts by occupation per one cycle and to assess the actual operation time of the machines. It should be stressed that, as stated before, it is the planned volume of work that is used in these estimates, that is, the rated figures multiplied by coefficients 0.9 or 0.85, depending on the length of the coal wall.

In the compiling of work schedules efficiency means *factual output per man-shift but not less than the set production quotas.* The achievement of the planned and, all the more so, rated face output would mean substantial overfulfilment of production quotas.

Besides miners, whose number per cycle is calculated as explained above, there are workers whose total number is determined by their distribution among the work places, in accordance with mining operations performed by each shift. These are cutting-machine and combine operators, their helpers, repairmen, blasters, motormen and chutemen.

This labour distribution should provide for the maximum mechanisation of odd and ancillary jobs, concurrent execution of several jobs by one man and elimination of idle time.

In general, the procedure to be adopted in distributing men for cyclic operation at a coal face is of particular importance.

In this way one can establish the total number of men actually engaged in a wall per cycle per day. In order to determine the total number on the *pay-roll*, the figure established should be multiplied by the coefficient 1.31 in the instance of a continuous work week and by 1.14 when the operation is carried out intermittently during the week.

5. *In assigning men and determining actual operation time of mine equipment* according to shifts, one should bear in mind the following.

A normal 24-hour duty cycle for a coal wall provides for two coal-turning and one back (repair and preparation) shifts. This work schedule for the walls is a very *important factor* of an operation based on one cycle per day. Two coal-getting shifts with the third devoted to preparatory work and repairs are necessary to prepare equipment and the work places.

Organisation of labour in a wall should also provide for possible combination of individual operations. At the same time projected organisation of work should not be too rigid, that is, it has to provide for extra time and efficiency so as to prevent any possible short delays in individual operations from deranging the succeeding ones and disrupting the whole cycle. In planning organisation of work, provision should be made for cutting power off for two or three hours during the back shift to ensure inspection of electric apparatus and instruments and repair of mains.

6. Lastly, the outlined organisation of work in the coal face is depicted graphically in *planned work diagrams and labour distribution charts*. These should be as simple as possible and distinct, since it is not only the members of the technical staff but miners too who should be familiar with them. Typical specimens of diagrams and charts are described below.

7. Graphic representations are supplemented by the principal *technical and economic indices*.

For each coal wall worked on the basis of cycles they should be as follows:

a) geological conditions (see above);

b) mining conditions: length of the wall; face advance per cycle; specified drilling and blasting operations; specified timbering; method of roof control;

c) mechanisation of the coal face: types of machines and their capacities;

d) organisation of work is characterised by planned work diagrams and labour distribution charts;

e) operative and economic results: coal output per cycle (in 24 hours); number of cycles per month; coal tonnage obtained from a wall in a month (planned); coal output per man in one shift and in a month, in tons; consumption of mine support materials—timber in cubic metres and metal in kilogrammes per 1,000 tons of coal extracted; consumption of explosives in grammes per ton of coal; prime cost of one ton of coal in the wall.

These indices are also necessary for compiling analogous indices *for the whole of a mine section*, which may include not one but two and even several coal walls and stopes in development workings. The cost of deliveries to the haulageway is likewise taken into account in this table.

Planned work diagrams and labour distribution charts are usually supplemented by a table of technical and economic indices which, for the sake of brevity, does not contain all of the above-cited indices but only the most important (Figs 185-187).

8. A feature characteristic of a mine face is its *mobility*. Hence, conditions prevailing in a coal wall change with the passage of time. For instance, when the angle of pitch in a slightly inclined seam decreases by a few degrees, the wall becomes appreciably longer. Consequently, all the earlier compiled work schedules and graphs may prove unsuitable and should be revised in good time. Such revision of work schedules should be done at the beginning of the month. Planned work diagrams and labour distribution charts are compiled in advance by the mine section superintendent for each individual coal wall in conformity with the general mine schedule and are approved by the chief engineer of the mine not later than five days before the end of each month. After their approval the schedules and graphs are made known to mine foremen, shift bosses and facemen.

## 13. Examples of Cyclic Work in Coal Walls

1. *24-hour cycle in a long wall.* This example refers to the working of a slightly inclined seam, 1.3 metres thick at the Lutugin Mine (Donets coal fields). The coal wall is 210 metres long. Coal is undercut by two coal cutters of the MB-60 type and then blasted out. Holes are drilled with electric augers. Coal is brought out of the wall by four CKP-11 chain-and-flight conveyers. Metal posts of СГК-2 type, wedged against cap boards, are used to support the face. Roof control is effected by building pack walls.

Operations in the wall are organised in the following manner (see Fig. 185).

By the beginning of the morning shift the face in the lower portion of the wall is undercut over a distance of 105 metres and holes in the coal have been shot. In the upper portion of the wall drill-holes in the lateral entries (dummy roadways) have already been blasted. All the four conveyers are shifted to a new position. The operations effected in the morning shift include extraction of coal in the lower portion of the wall and delivery of mine timber. In the upper portion of the wall the operations include undercutting of coal, drilling and shooting of holes in coal and arrangement of pack walls. To preclude any delays in undercutting, pack walls are built from 140 metres. At first only two men are engaged at each pack wall and they build four lower pack walls. After the lower rock walls have been backed up to three-quarters of their height, the fillers leave the lower rock wall one by one and go over to packing the upper ones. Having built the lower rock walls, the four fillers help to complete the upper ones.

The operations in the second shift include extraction of coal in the upper section of the wall, delivery of mine timber and drilling of holes in dummy roadways over the entire length of the coal wall.

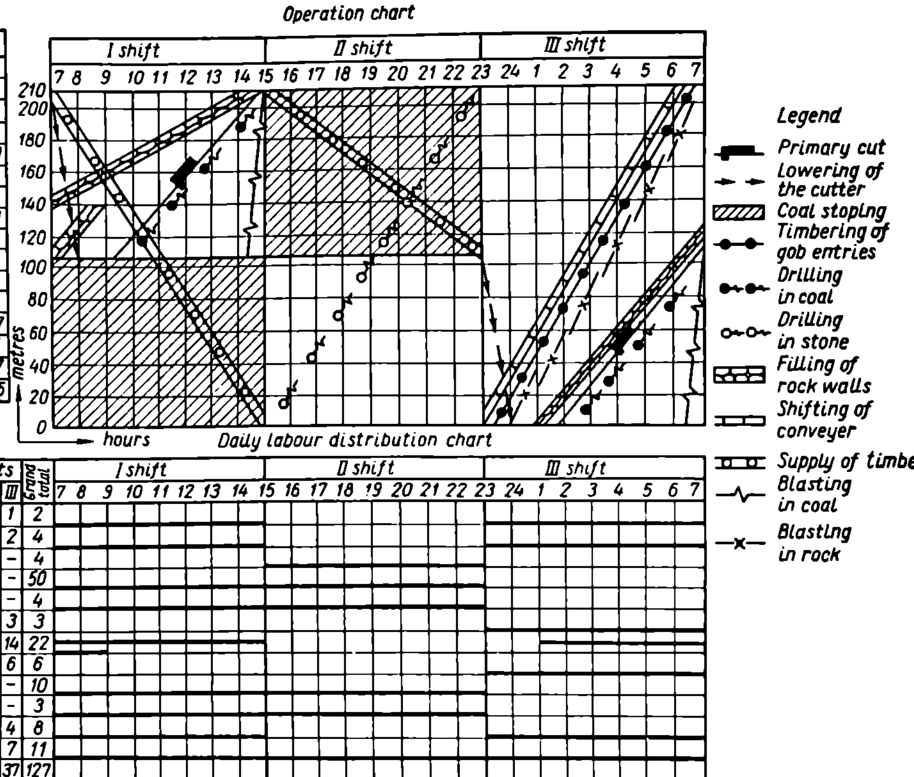
The night, back, shift is used up for moving conveyers to a new position, timbering dummy roadways and building pack walls. In the lower section of the wall the cutting machine is lowered to its initial position and holes in coal are drilled and shot. When needed, mine equipment throughout the coal wall is inspected and repaired.

Distribution of labour and the ultimate results of mining operations are illustrated in Fig. 185.

2. *One cycle per 24 hours with a mine combine* (Fig. 186). The work schedule, labour distribution chart and table of technical and economic indices are compiled for the following conditions: the coal seam mined is a gently inclined (10°) one, 0.95 metre thick, yielding 1.22 tons per sq metre of the seam surface. The coal wall extends over 150 metres. Coal is mined by a Donbas cutter-loader (combine). The undercutting and breaking operations, loading of coal onto the conveyer and its transportation from the wall follow each other in a continuous flow, and this makes the planned work diagram quite simple.

As can be seen from the work schedule, the combine extracts and loads coal during the morning and day shifts, moving upwards. It is lowered during the night shift. Before its operation special "niches" are blasted out and coal is loaded onto the conveyer from face sections, each seven metres long (see hatched areas in Fig. 186). Mine timber is supplied to the face from above. As in the previous example, to obtain partial filling twelve dummy roadways (lateral entries) are driven immediately behind the advancing face, furnishing sufficient waste for six-metre-wide pack walls. The upper portion of the wall is packed during the morning shift, the lower in the night. During the night shift holes are drilled and at the beginning and the end of the morning shift explosive charges are shot in the rock of lateral entries. Occupational groups, numbers and distribution of workers according to shifts, are presented in Fig. 186 together with the technical and economic indices.

| Technical and economic indices | |
|---|---|
| Extent of longwall, m | 210 |
| Seam thickness, m | 1.30 |
| Angle of pitch, degrees | 10 |
| Cutting depth, m | 1.8 |
| Recovery rate per 1 $m^3$ of seam area, t | 2.09 |
| Coal output per operation cycle, t | 789 |
| Number of cycles per day | 1.0 |
| Daily coal production program, t | 670 |
| Roof control | Partial back filling |
| Number of rock walls | 20 |
| Width of rock wall, m | 7.0 |
| Man-shifts per day | 127 |
| Planned average daily output per man-shift, t | 5.27 |
| Total number of miners | 166 |
| Planned monthly output per stoper, t | 1.37 |
| Cycle rate | 25.5 |

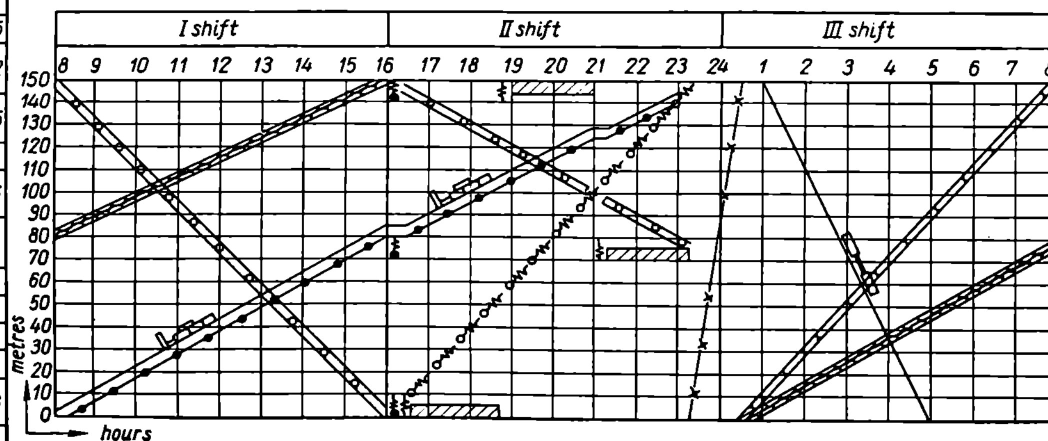


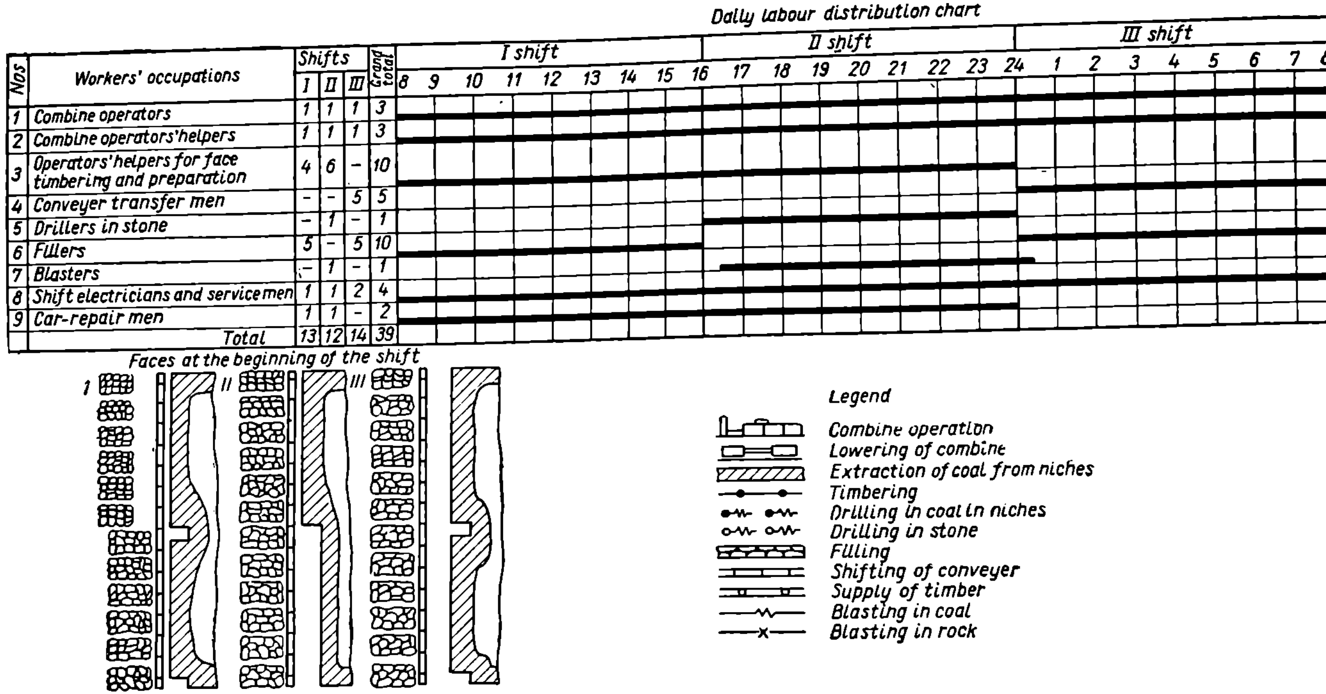

| Class of work | Shifts I | Shifts II | Shifts III | Grand total |
|---|---|---|---|---|
| Cutting machine operators | 1 | – | 1 | 2 |
| Operators' helpers | 2 | – | 2 | 4 |
| Drillers in stone | – | 4 | – | 4 |
| Face-men | 25 | 25 | – | 50 |
| Chute pullers | 2 | 2 | – | 4 |
| Timbermen in gob entries | – | – | 3 | 3 |
| Fillers | 8 | – | 14 | 22 |
| Conveyer transfer men | - | – | 6 | 6 |
| Motormen | 5 | 5 | – | 10 |
| Timber supply men | 2 | 1 | – | 3 |
| Blasters and their helpers | 4 | – | 4 | 8 |
| Shift electricians and service men | 2 | 2 | 7 | 11 |
| Total | 51 | 39 | 37 | 127 |

*Fig. 185.* Work schedule diagram for a long coal wall

| Nos | Technical and economic indices | | Un. of meas. | Value |
|---|---|---|---|---|
| 1 | Extent of longwall | | m | 150 |
| 2 | Seam thickness | | m | 0.95 |
| 3 | Angle of pitch | | degrees | 10 |
| 4 | Cutting depth | | m | 1.45 |
| 5 | Recovery rate per 1 $m^2$ of seam area | | t | 122 |
| 6 | Coal output per operation cycle | | t | 265 |
| 7 | Number of cycles per day | | – | 1 |
| 8 | Daily coal production program | | t | 235 |
| 9 | Percent of mechanised loading | | % | 85 |
| 10 | Roof control | Partial backfilling | | |
| 11 | Number of rock walls | | – | 12 |
| 12 | Width of rock walls | | m | 6 |
| 13 | Man-shifts per day | | – | 39 |
| 14 | Output per man-shift (according to the chart) | | t | 6.8 |
| 15 | Total number of miners | | men | 51 |
| 16 | Monthly output per stoper | | t | 138 |

Operation chart





Daily labour distribution chart

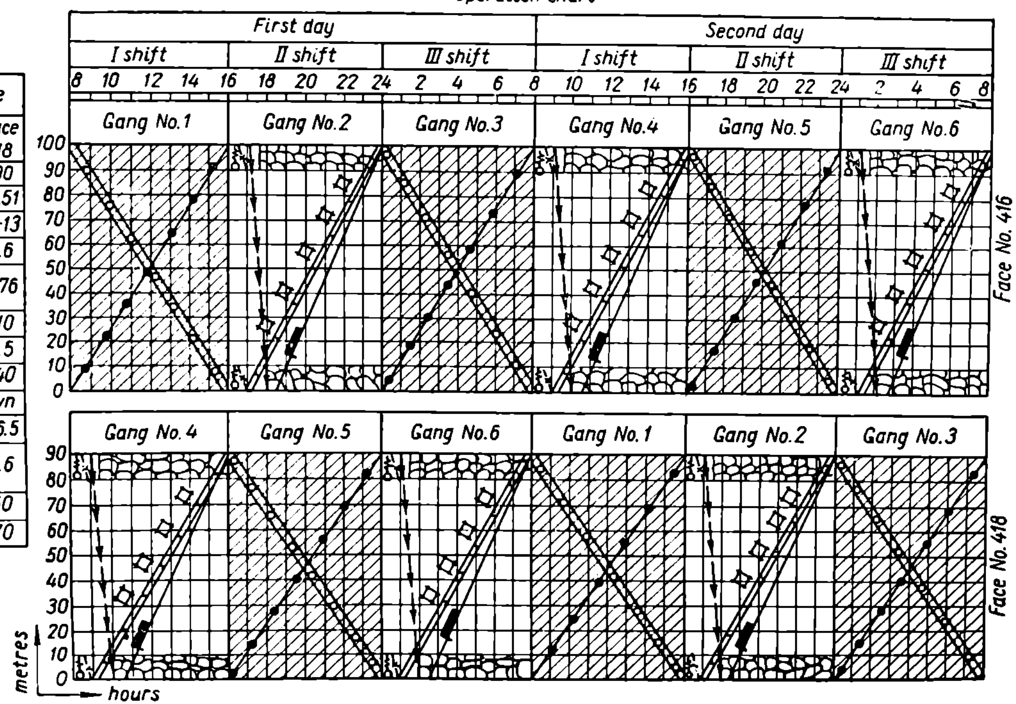
| Nos | Workers' occupations | Shifts I | Shifts II | Shifts III | Grand total |
|---|---|---|---|---|---|
| 1 | Combine operators | 1 | 1 | 1 | 3 |
| 2 | Combine operators' helpers | 1 | 1 | 1 | 3 |
| 3 | Operators' helpers for face timbering and preparation | 4 | 6 | – | 10 |
| 4 | Conveyer transfer men | – | – | 5 | 5 |
| 5 | Drillers in stone | – | 1 | – | 1 |
| 6 | Fillers | 5 | – | 5 | 10 |
| 7 | Blasters | – | 1 | – | 1 |
| 8 | Shift electricians and service men | 1 | 1 | 2 | 4 |
| 9 | Car-repair men | 1 | 1 | – | 2 |
| | Total | 13 | 12 | 14 | 39 |

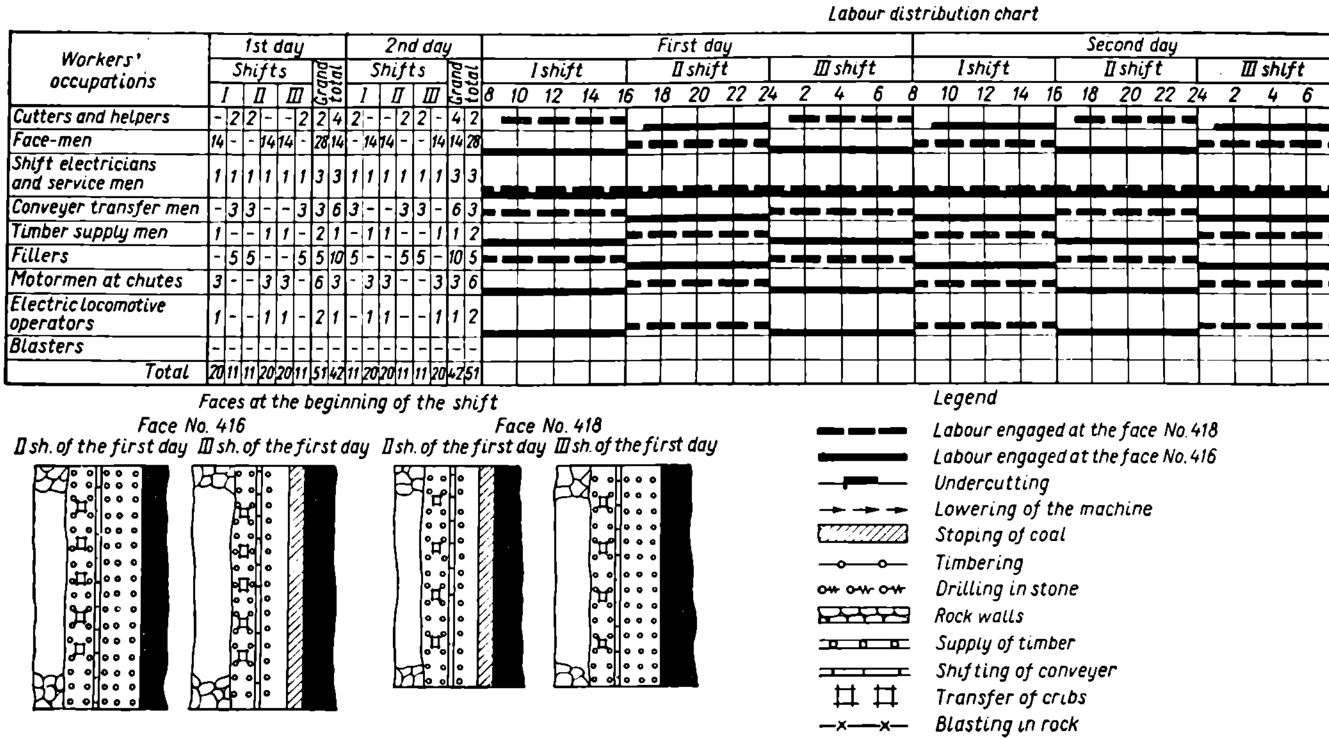


*Fig. 186.* Work schedule diagram for one cycle in 24 hours with the extraction of coal by the Donbas combine

| Nos | Technical and economic indices | Un. of meas. | Value Face 416 | Value Face 418 |
|---|---|---|---|---|
| 1 | Extent of longwall | m | 100 | 90 |
| 2 | Thickness of seams | m | 0.5 | 0.51 |
| 3 | Angle of pitch of seams | degrees | 7-13 | 7-13 |
| 4 | Cutting depth | m | 1.6 | 1.6 |
| 5 | Recovery rate per 1 m² of seam area | t | 0.75 | 0.76 |
| 6 | Coal output per operation cycle | t | 120 | 110 |
| 7 | Number of cycles per day | | 1.5 | 1.5 |
| 8 | Daily coal production program | t | 155 | 140 |
| 9 | Roof control | Gradual settling down | | |
| 10 | Man-shifts per day | - | 46.5 | 46.5 |
| 11 | Output per man-shift (according to chart) | t | 3.9 | 3.6 |
| 12 | Total number of miners | men | 60 | 60 |
| 13 | Monthly output per stoper | t | 78 | 70 |

Operation chart






| Workers' occupations | 1st day | | | | | | | | 2nd day | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Shifts | | | | | | Grand total | | Shifts | | | | | | Grand total | |
| | I | | II | | III | | | | I | | II | | III | | | |
| Cutters and helpers | - | 2 | 2 | - | - | 2 | 2 | 4 | 2 | - | - | 2 | 2 | - | 4 | 2 |
| Face-men | 14 | - | - | 14 | 14 | - | 28 | 14 | - | 14 | 14 | - | - | 14 | 14 | 28 |
| Shift electricians and service men | 1 | 1 | 1 | 1 | 1 | 1 | 3 | 3 | 1 | 1 | 1 | 1 | 1 | 1 | 3 | 3 |
| Conveyer transfer men | - | 3 | 3 | - | - | 3 | 3 | 6 | 3 | - | - | 3 | 3 | - | 6 | 3 |
| Timber supply men | 1 | - | - | 1 | 1 | - | 2 | 1 | - | 1 | 1 | - | - | 1 | 1 | 2 |
| Fillers | - | 5 | 5 | - | - | 5 | 5 | 10 | 5 | - | - | 5 | 5 | - | 10 | 5 |
| Motormen at chutes | 3 | - | - | 3 | 3 | - | 6 | 3 | - | 3 | 3 | - | - | 3 | 3 | 6 |
| Electric locomotive operators | 1 | - | - | 1 | 1 | - | 2 | 1 | - | 1 | 1 | - | - | 1 | 1 | 2 |
| Blasters | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Total | 20 | 11 | 11 | 20 | 20 | 11 | 51 | 42 | 11 | 20 | 20 | 11 | 11 | 20 | 42 | 51 |

*Fig. 187.* Work schedule diagram for 45 cycles per month (three cycles every 24 hours, in two coal walls)

3. In Section 9 we said that engineer Kosenko had proposed *a 45-cycle operation schedule* in coal walls in certain conditions. Fig. 187 illustrates an example of such organisation of work. It refers to mining two closely situated coal walls at the Vorovsky Mine in the Donets coal fields, where two anthracite beds are being worked, with total thickness of 0.6-0:7 metre each and working thickness of 0.45-0.55 metre. Each wall is 90-100 metres long and anthracite is undercut by machines. These two coal walls are worked by nine teams: three gangs of loaders, three complex preparatory teams and three cutting-machine operators. The teams are engaged now in one and now in another wall, in the order shown in Fig. 187.

Mining operations in each wall are thus repeated every eight hours, and all the members of the teams, both those mining coal and preparing walls, work in the same shifts throughout the month. According to the work schedule, one cycle is accomplished in each wall during the first 24 hours and two in the next 24 hours, thus making it 45 cycles per month.

In spite of the thinness of the seam, the walls yield 5,500-7,000 tons of coal a month each, while output per one man per shift comes to 3.9-4.8 tons.

## 14. Prerequisites for Successful Fulfilment of Cyclic Operations

The fulfilment of the cyclic operation outlined and plotted in the planned work diagram and labour distribution chart requires a number of technical and organisational conditions, both at the production face and outside it.

I. The following prerequisites are of importance *technically*.

1. The coal face should be *rectilinear*, without bends. This is required for the efficient operation of coal cutters and cutter-loaders, as well as for the normal operation of conveyers. If the face is badly curved, the combine or coal cutter may veer off its breast, and that would diminish the depth of the cut. In a bent face it is impossible to stretch out the rope of the cutting machine to any great length and it is necessary frequently to change the position of the anchor post. No belt or chain-and-flight conveyer can be employed in a non-rectilinear face. A straight-lined production face is also of importance for efficient roof control when mining involves artificial caving.

It is not the technical staff of the mine section alone that has to see to it that the face remains straight. This is also the duty of the mine surveyor, who is responsible for the shape of the coal face.

2. In selecting the *roof-control* method, one should take account of the properties of wall rocks. When it is the method of mining with

fill that is adopted, the packing should not fall behind the advancing face more than stipulated by the specifications for mine support. When roof control is effected by artificial caving, one should keep to the space interval or run of caving confirmed by practice.

3. The face should be *timbered* in accordance with the accepted specifications, and the supporting sets should be put up in good time. Metal or timber props should be arranged along the coal face in straight rows. The condition of the roof should be constantly looked after and all the sections exciting apprehension should be immediately reinforced. A reversible-type chain conveyer can also deliver the mine timber to the coal wall via a lower-lying entry. The question of mine support stocks is discussed below.

4. The coal wall should be *equipped with machinery*—mine combines, coal cutters and loaders, air hammers and conveyers with rated capacities sufficient to achieve the goals set by planned work schedules.

The length of the cutter bar should accord with the properties of the seam, and the cutting machine itself should be operated properly. The coal cutter's teeth should be changed in good time. For harder grades of coal they should be faced with hard alloys. The track for the coal cutter or mine combine should be kept free of coal or rock debris and the coal face should be smooth and have no benches or overhangings which might hinder the machine's motion.

The holes drilled in coal should not be shallower than the cuts, and they should be properly spaced. The loaders should break off the undercut coal over the entire area of the breast, leaving the face in the shape of a vertical surface from top to bottom. Moreover, if the machine, because of the seam structure, cuts somewhat above the bottom of the bed, the workers should break off all the coal lying below the cut (bottom coal).

Maintenance of face machinery and other electrical and mechanical equipment is discussed below.

5. The coal wall should be well *ventilated*, in accordance with the regulations. An effective air current is also essential for reducing the time needed for blowing out the noxious gases created by the explosion of charges in holes drilled in coal and dummy roadways. The best time for blasting is between two shifts. Since coal cutters and loaders are apt to produce much coal dust, the wall should be equipped with spraying devices.

6. Proper *lighting* at the face is not only necessary to ensure safety; it also helps increase labour efficiency. All the loading stations near the walls should have electric lighting.

7. The coal wall should be connected by telephone to the mine despatcher's office. For convenience's sake, the telephone should be installed at the loading station of the wall.

II. The following points are important for the *organisation* of work:

1. Mining teams or gangs should be formed as stipulated in the planned work schedule and labour distribution chart, both with regard to the number of men and their qualification.

2. *Complex shiftwise teams* are the best form of labour organisation.

In coal walls serviced by cutter-loaders, a mining shift team is composed of the cutter, his assistant, and helpers in timbering and shaping the face.

In walls worked by coal cutters, a complex team is made up of loaders, timberers, drillers and timber-supply men.

A shift team for moving conveyers consists of conveyer transfer men (fitters) and a shift electrician.

A coal-cutter team of the repair-preparatory (back) shift has a coal-cutter operator, one or two helpers and a shift electrician.

Roof control in coal walls is effected by gangs made up of fillers, rock drillers, rib- or break-line timbermen, cribmen and roofmen (prop-pulling men).

Every team has a leader.

Workers not included in the above-mentioned teams are assigned individual work places and jobs in accordance with the labour distribution chart.

3. Each face man should be acquainted with the planned work schedule and labour distribution chart and know his work place. To this end, he must be appropriately instructed by the mine-section superintendent, his assistants and team leaders. Each miner is familiarised with the labour distribution chart for the month a few days before it comes into force.

4. In the case of operations proceeding uninterruptedly, the miners of the succeeding shift relieve their colleagues right *at their work place*.

5. Accomplished tasks are *checked at the work places* by the mine-section supervisory staff after each shift.

6. Efficient work by each miner, in addition to proper distribution of labour and employment of suitable equipment, is a prerequisite for the fulfilment and overfulfilment of the coal production programme.

The best possible organisation of work and employment of any highly efficient device will be of no avail if, because of a poorly done job, the schedule is not adhered to and the device is inadequately utilised. Therefore, the members of a team should abide strictly to labour discipline, bearing in mind that laggard often holds back the others.

The slightest inrush of roof rocks on account of inadequate timbering delays the cutting of coal, and this will affect all the successive mining operations.

Besides abiding strictly to the work regulations, the members of a team should all be highly conscientious. Their basic aim should be maximum efficiency and best possible utilisation of mine equipment.

These aims are achieved by:

a) skill and ability to make proper use of the existing conditions in attaining maximum efficiency;

b) full use of shift time;

c) assiduity;

d) economy of time through the elimination of unnecessary interruptions in work;

e) proper maintenance of equipment (running repairs, timely lubrication and cleaning, change of worn-out parts, etc.);

f) concern for both the quantity and quality of coal extracted; each miner should keep down the ash content.

Good work brings high wages and better living conditions, which, in turn, help further to raise efficiency.

7. The rates of wages in complex teams are worked out on the basis of the following units: in mining shift gangs—per ton of coal extracted; in teams engaged in shifting conveyers—per metre; in a roof-control gang—per square metre of the stope area, measured once a month by the surveyor.

III. Apart from adequate organisation of work in the production face, successful winning of coal depends also on the following *conditions*:

1. The face should have a *continuous* supply of electric power and, when necessary, of *compressed air*.

2. *Mine timber* of proper quality and size should be supplied in good time.

3. Adequate provision should be made for *preventive inspection and repair* of mine equipment and mechanisms, as specified in a special schedule.

A mine section should have a plan for maintenance of equipment, compiled in accordance with standards set for repairs of mine equipment, regulations for technical exploitation of mines, and the cyclic work schedule. Running repairs are effected during the back (preparatory-repair) shift. There should be a sufficient stock of rapidly wearing parts and fitter's tools available. For major repairs and overhauls the machines are brought to the surface and sent to mine machine shops.

4. Provision should be made for adequate transportation of coal drawn from the production faces. This requires a sufficient stock of empty mine cars in good repair always available at loading stations, electric locomotives of needed power capacity, haulageways and mine tracks in good condition.

5. Efforts should be made to prevent development work from delaying the normal course of stoping operations in any way. In breast stoping, particularly, the heading of the haulageway or entry should always be driven in advance of the coal faces. The rate of this advance should allow switching operations at the loading stations near the coal walls.

6. The *airway* or *air entry* should be kept in good condition to allow not only free passage of the air current, but also to provide for convenient delivery of mine timber to production faces.

7. Implementation of the cyclic work schedule is *supervised* not only by team leaders and mine foremen but also by mine-section superintendents and their assistants. If they are not present in a coal wall at any given moment, they should keep in touch with it by telephone and come over when required to check on the implementation of the cyclic work schedule and, if necessary, take measures to ensure uninterrupted mining operations.

8. Control over the actual fulfilment of planned cyclic operations and registration of their results require *working face records.*

A working face record (Fig. 188) includes the following operative data for every workday:

a) actual state of mining operation in the wall at the end of each shift. There, conventional signs show the amount of coal undercut, blasted and extracted along the length of the wall and the position of mining machines (cutting, loading and mine combines) and conveyers. Authenticity of the information is testified to by the signatures of the shift manager on duty and the mine foreman;

b) statement on factual labour distribution according to shifts and classes of work;

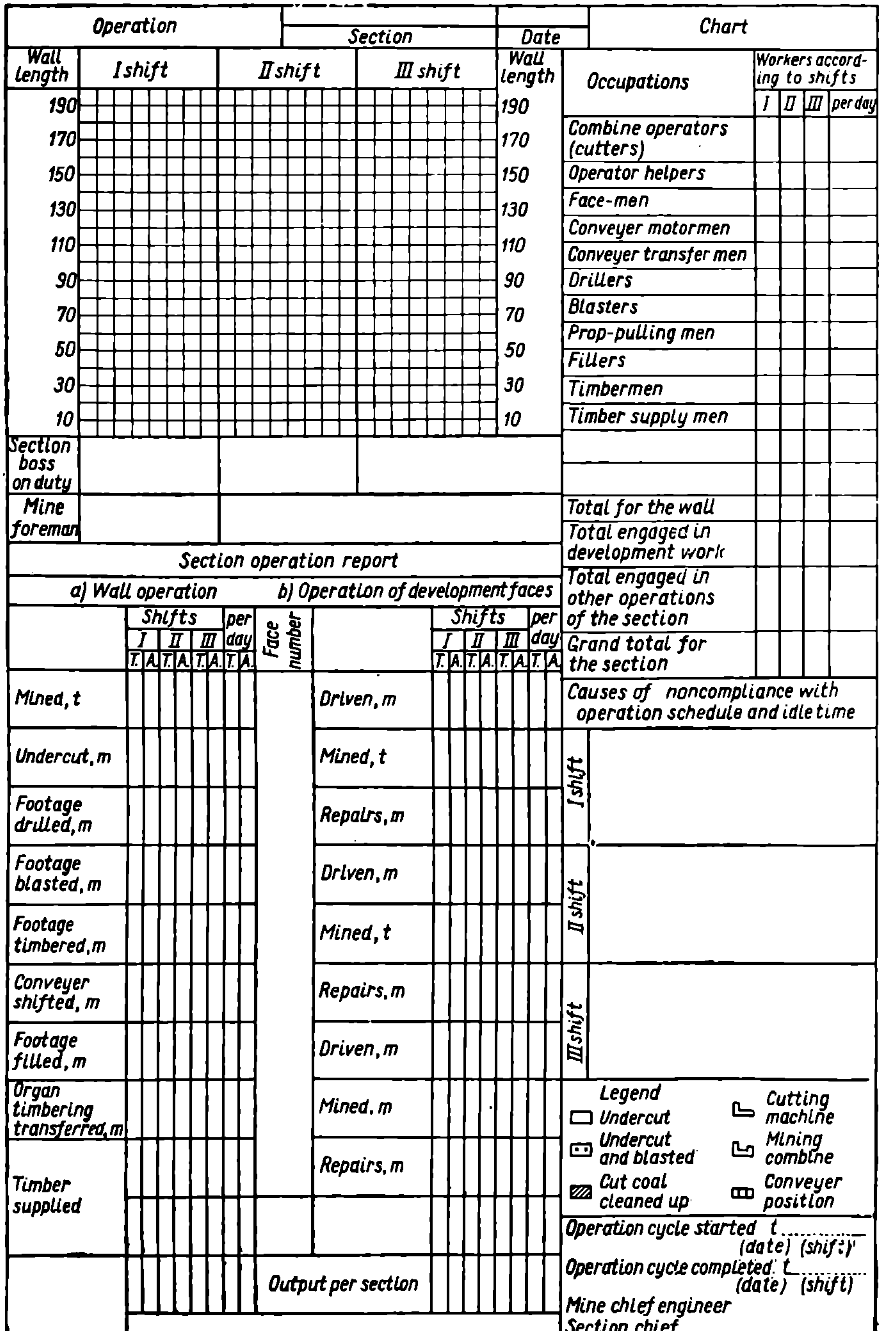
c) condition of the roof all along the coal wall, according to shifts;

d) entry in the face record of all possible instances of noncompliance with the cyclic schedule of work, periods of idle time and the causes underlying them;

e) finally, entry in the face record of assignments and accomplished work per shifts and day—actual coal output, undercutting, drilling, timbering, conveyer shifting, etc.

The forms of the working-face records are filled out by the mine-section superintendent or his assistant on the basis of information contained in the shift manager's report at the end of each working shift (Fig. 188), or in the mine despatcher's report (Fig. 189).

9. The working-face records also serve for registering the results of cyclic operations performed in a day, week or ten days, or month. The data they contain are checked and compared every ten days and every month with the results of the surveyor's measurements of the face advances and with current information on the amount of coal actually extracted.



Operation
Section
Date
Chart
Wall length
I shift
II shift
III shift
Wall length
190
170
150
130
110
90
70
50
30
10
Occupations
Workers according to shifts
I
II
III
per day
Combine operators (cutters)
Operator helpers
Face-men
Conveyer motormen
Conveyer transfer men
Drillers
Blasters
Prop-pulling men
Fillers
Timbermen
Timber supply men
Section boss on duty
Mine foreman
Total for the wall
Total engaged in development work
Total engaged in other operations of the section
Grand total for the section
Section operation report
a) Wall operation
b) Operation of development faces
Shifts
per day
I
II
III
T.
A.
Face number
Mined, t
Undercut, m
Footage drilled, m
Footage blasted, m
Footage timbered, m
Conveyer shifted, m
Footage filled, m
Organ timbering transferred, m
Timber supplied
Driven, m
Mined, t
Repairs, m
Driven, m
Mined, t
Repairs, m
Driven, m
Mined, m
Repairs, m
Output per section
Causes of noncompliance with operation schedule and idle time
I shift
II shift
III shift
Legend
Undercut
Undercut and blasted
Cut coal cleaned up
Cutting machine
Mining combine
Conveyer position
Operation cycle started t............
(date) (shift)
Operation cycle completed t............
(date) (shift)
Mine chief engineer
Section chief

*Fig. 188.* Working face cyclic operations record

| Dispatcher's | Section | Face number | Date | Operation chart |
|---|---|---|---|---|

| Operations \ Time | | I shift | | | | II shift | | | | III shift | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | 9 | 11 | 13 | 15 | 17 | 19 | 21 | 23 | 1 | 3 | 5 | 7 |
| Operation and lowering of the mining combine or cutting and lowering of the cutting machine | 150 | | | | | | | | | | | | |
| | 130 | | | | | | | | | | | | |
| | 110 | | | | | | | | | | | | |
| | 90 | | | | | | | | | | | | |
| | 70 | | | | | | | | | | | | |
| | 50 | | | | | | | | | | | | |
| | 30 | | | | | | | | | | | | |
| | 10 | | | | | | | | | | | | |
| 1. Coal unloading (with hand operation) or coal mining from niches 2. Blasting in coal 3. Conveyer shifting | 150 | | | | | | | | | | | | |
| | 130 | | | | | | | | | | | | |
| | 110 | | | | | | | | | | | | |
| | 90 | | | | | | | | | | | | |
| | 70 | | | | | | | | | | | | |
| | 50 | | | | | | | | | | | | |
| | 30 | | | | | | | | | | | | |
| | 10 | | | | | | | | | | | | |
| 1. Filling or caving 2. Drilling in stone 3. Blasting in rock 4. Timbering | 150 | | | | | | | | | | | | |
| | 130 | | | | | | | | | | | | |
| | 110 | | | | | | | | | | | | |
| | 90 | | | | | | | | | | | | |
| | 70 | | | | | | | | | | | | |
| | 50 | | | | | | | | | | | | |
| | 30 | | | | | | | | | | | | |
| | 10 | | | | | | | | | | | | |

8 10 12 14 16 18 20 22 24 2 4 6 8

| Rate of coal loading per hour | | |
|---|---|---|
| Faces | Task | |
| | Actual | |
| Other slopes of the section | Task | |
| | Actual | |
| Grand total for the section | Task | |
| | Actual | |

| Labour engaged in the operation | | | | |
|---|---|---|---|---|
| Classes of work | Number of workers according to shifts | | | |
| | I | II | III | per day |
| Combine operators (cutter runners) | | | | |
| Operator helpers | | | | |
| Face-men | | | | |
| Motormen at the conveyer | | | | |
| Conveyer transfer men | | | | |
| Drillers | | | | |
| Blasters | | | | |
| Prop-pulling men | | | | |
| Fillers | | | | |
| Timbermen | | | | |
| Timber supply men | | | | |
| | | | | |
| | | | | |
| | | | | |
| | | | | |
| | | | | |
| Total labour at the face | | | | |
| Total labour in development work | | | | |
| Total labor engaged in other operations of the section | | | | |
| Total for the section | | | | |

Section operation report

a) Wall operation

| | Shifts I T | I A | II T | II A | III T | III A | per day T | per day A |
|---|---|---|---|---|---|---|---|---|
| Mine, t | | | | | | | | |
| Undercut, m | | | | | | | | |
| Footage drilled, m | | | | | | | | |
| Footage blasted, m | | | | | | | | |
| Footage timbered, m | | | | | | | | |
| Conveyer shifted, m | | | | | | | | |
| Footage filled, m | | | | | | | | |
| Organ timbering transferred, m | | | | | | | | |
| | | | | | | | | |
| | | | | | | | | |
| Timber supplied | | | | | | | | |

b) Operation of development faces

| Face number | | Shifts I T | I A | II T | II A | III T | III A | per day T | per day A |
|---|---|---|---|---|---|---|---|---|---|
| | Driven, m | | | | | | | | |
| | Mined, t | | | | | | | | |
| | Repairs, m | | | | | | | | |
| | Driven, m | | | | | | | | |
| | Mined, t | | | | | | | | |
| | Repairs, m | | | | | | | | |
| | Driven, m | | | | | | | | |
| | Mined, t | | | | | | | | |
| | Repairs, m | | | | | | | | |

Name of the mine foreman

Dispatcher's signature

Sketched actual face position at the close of the shift

| I shift | II shift | III shift | Wall length |
|---|---|---|---|
| | | | 150 |
| | | | 130 |
| | | | 110 |
| | | | 90 |
| | | | 70 |
| | | | 50 |
| | | | 30 |
| | | | 10 |

Operation cycle started ............ (date) (shift)

Operation cycle completed ........... (date) (shift)

Mine chief engineer

Section chief

*Fig. 189.* Dispatcher's working face cyclic operations record

The working-face records are very important documents which, if kept properly and systematically, reflect all the cyclic operations effected in coal walls, their results, shortcomings and the causes underlying them.

### B. STEEPLY PITCHING SEAMS

## 15. Peculiarities of Rock Pressure and Support of Working Faces in Steeply Pitching Seams

We have already seen (Chapter II) that highly inclined seams are opened up by crosscuts (Fig. 190)—by haulage one *a* and ventilation one *b*, on both sides of which *main level entries ac* and *bd* are pushed forward. When the breasts of these entries are several scores of metres ahead of the crosscut they are connected by break-throughs *e* and *f*. Production faces can be started from these break-throughs.

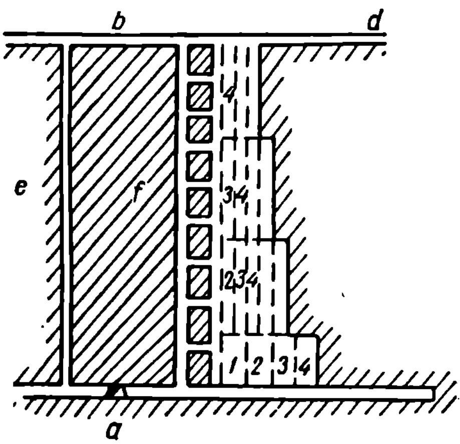



*Fig. 190.* Opening up and development of a steeply pitching seam

In the simplest case, *one rectilinear* continuous working face, similar to that in slightly inclined seams, can be established in a level. This type of mining is specific of the longwall system applied in steeply dipping beds. If, because of geological disturbances or insufficient stability of wall rocks, one continuous face should prove inconvenient, the level may, even in a steeply pitching seam, be divided into two or several sublevels, each mined through a single production face.

When the pitch is heavy, however, continuous (fully or approximately) dip faces are used only when coal is extracted by mine combines, coal cutters or coal ploughs (see below). The breaking off of coal by pneumatic hammers gives the faces the shape of an *overhand stope* and this is due to the following reasons.

When a lump of coal, a block or a rock falls down in a working face with a low pitch, it remains where it is. In a steeply pitching seam a fallen object rolls down. On its way down it may happen to hit other objects—props of support sets, etc.—and cause them to fall, and that may lead to further caving of rocks and even to an accident

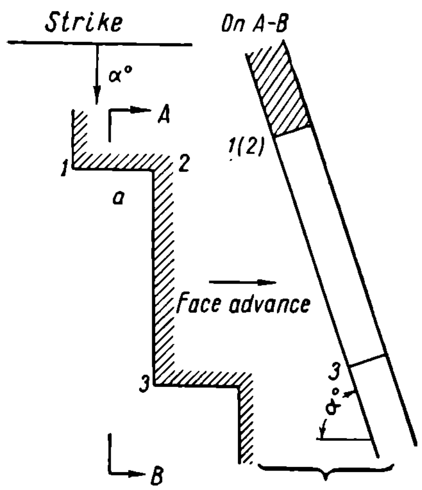



*Fig. 191.* Outline of an overhead face

in the face. For this reason it is necessary to pay particular attention to the support of the active stope area.

In a straight continuous dip face falling objects roll down directly into the area in the vicinity of the face, and that is why special safety measures should be taken in mining such faces (see below). Therefore, when coal is extracted with pneumatic hammers production faces in steeply inclined beds are subdivided into *benches* (see Fig. 191).

The face *2-3* of each bench is called the *breast* of the bench, while surface *1-2*, overlying it, the *overlap* of the bench. Line *2-3* is called the bench *height*, while overlap *1-2* defines the advance of the lower bench over the top. Angle *2* is usually referred to as the "corner" of the bench.

Coal broken off in each bench rolls down along the line of pitch without hitting the men in the lower benches. The purpose of the overhand stope is thus to *safeguard* men from falling coal (or other objects) excavated in the benches above.

Fig. 190 shows how an overhand or overhead stope is gradually cut until it is extended over the whole of the level interval (or that of sublevel). The figures indicate coal areas worked in successive shifts. At first one man mines out face area *1*; during the next shift two facemen work out two areas *2*, *2* and so on.

The coal loosened in the benches, sliding along the bottom of the seam, falls directly onto the lower portion of the face, or the floor boards laid along the bench line. This facilitates its reaching the lower section of the level where it is loaded into the mine cars through chutes. The mined-out space is filled either completely or partially, or not at all. In the first instance, the stub pillars of coal along the entries may be either left or not, but in the second and third instances they must always be left, except for the rare occasions when they can be replaced by pillars of blasted waste rock.

Reliable support for the mined-out area in steeply dipping beds is also of particular importance because there is not only the possibility of the hanging-wall rocks caving in, but also that of the foot-wall rocks sliding down.

Active faces are supported by runs of posts following the line of dip. It is only in the case of extremely firm wall rocks that posts are set up directly in hitches cut in the floor and blocked against the roof. Ordinarily, slabs *a* are first arranged on the hanging and foot walls (Fig. 192) and the posts are then driven in between them.

To prevent timbering from sliding the slabs should meet *end-to-end* (and not overlap each other as is the case in gently sloping beds). The posts should be placed at the ends of each slab and, in addition to that, one, two or three supplementary posts should be put up between the end props, this depending upon the properties of wall rocks and the length of slabs.

Spacing between the rows of props varies, usually from 0.7 to 1 metre, depending on the stability of hanging and foot-wall rocks. In the instance of weaker wall rocks prop and slab support is insufficient, inasmuch as individual blocks of rock can fall from between the rows of timber sets. In such cases short slabs (laggings) (*b* in Fig. 192), are driven under the main slabs. The number of laggings depends on the properties of rocks.

Since the overlap of each bench lies directly above the men working in the bench below, it should be secured and lagged with slabs.

*Cribbing* is often employed in the stoping area. The cribs are built on the sticks of usual sets (Fig. 193*a*), or else two additional props are set up at the bottom of the crib (Fig. 193*b*).

In steeply inclined beds the delivery of mine timber to the production faces is a difficult job. For the top benches the props and slabs are brought up in mine cars through the airway; for lower ones through the lower haulageway. To deliver mine timber to benches of low height, the facemen take their places in every bench and relay the

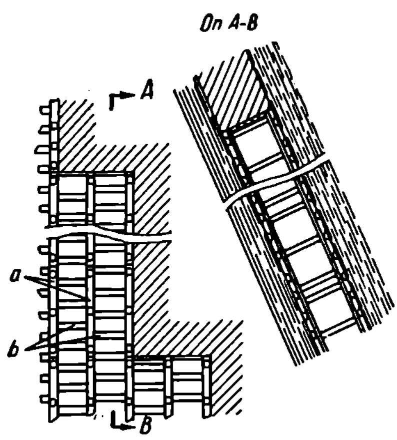



*Fig. 192.* Bench timbering

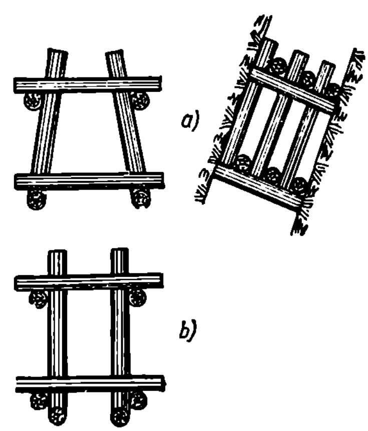



*Fig. 193* Cribbing in steeply pitching beds

props, slabs and laggings in numbers required for work during the shift. These materials are piled on support posts in front of each bench. In benches of considerable height it is possible to lower mine timber in a special box with runners or skids with the aid of a hand-operated or mechanised winch set up in the upper entry.

As said above, in steeply inclined beds both roof fall and bottom slide are possible, and falling blocks of coal or rock roll down unhindered. This makes adequate support of the production face and wall-rock control especially important. We have already seen that with the caving methods of mining in gently sloping seams it is of particular importance for systematic roof control to knock down the posts of the face timbering regularly and move special support sets at preset intervals. But in the case of a heavy pitch all this is actually ruled out, for here it would be rather unsafe to knock down the props. Here cribs are moved only occasionally. That is why, in the case of mining steeply inclined beds with partial or complete caving, protection of working areas from rock falls provides for careful arrangement of mine supports and systematic building of cribs, whose number and interspacing are determined on the basis of previous experience. As a rule, the cribs are abandoned in the worked-out area. In other words, when mining steep seams with partial or complete caving, wall-rock control is effected by retreat from rock collapses or goafs. Besides, in order to secure the stability of wall rocks it is sometimes necessary to leave small *pillars* (stumps) *of coal* three to five metres wide and high in worked-out areas. The abandonment of stumps increases coal losses and tends to complicate the delivery of coal in the stope area. The stumps are left somewhat below the centre line of the worked-out area.

In conditions under discussion the most reliable method of lending stability to wall rocks is by *filling the mined-out space.*

Packing materials needed in the mining of thin and medium-thick steep seams can be obtained both *underground* and *on the surface.* In the case of the former, they include interlayers or rocks of the false bottom or roof in the working seam itself and rocks obtained from newly driven or retimbered mine workings; in the latter case, from quarries, concentration plants (tailings) and old rock dumps.

Heavier partings or intercalations met with during the extraction of coal in benches are cut out separately and thrown over the floor boarding into the goaf to pack it.

To let down the rocks of partings and thin bands contained in the false bottom or roof, the usual practice is to tear off one or two lower boards of the flooring before cutting out the rock. The lumps of rock then fall through the flooring into the goaf. The volume of rock obtained from intercalations and the false roof or bottom (if they are thin enough to descend by themselves when coal is broken off) is nearly always insignificant, and cannot be regarded as a major source of mine-fill. These rocks can be thrown over the floor boarding only if there is no danger of their self-ignition in the goaf.

There is much more packing material to be obtained from the brushing of entries. If the levels are not subdivided, filling material can be obtained by driving a lower haulageway or an upper ventilation entry. It is rather difficult to employ a filling material obtained as the result of slashing in the lower entry because this requires either building rock sill pillars over it, which is not always practised because of the high cost, or else tramming mine cars with waste along the lower haulageway towards the crosscut and the shaft, hoisting it up to the ventilation level and tramming along the ventilation crosscut and airway to the stoping area and, finally, lowering it into the mined-out space. Barren rocks obtained in the process of driving the lower entry and as the result of its repair are therefore not used for filling, but brought to the surface and disposed of in waste dumps.

Appreciable amounts of packing material are obtained from the upper airway. Except for the uppermost level, this entry, as a rule, is pushed forward through the former haulageway goaf, and that is why its drifting yields relatively large amounts of waste. It is much simpler to use this waste for filling, for the distance over which it has to be transported from the entry heading to the benches is short and at the point of destination the mine cars can be discharged by a tipper or directly through a side door into the worked-out area. The same method is employed to deliver waste obtained from the repairs of the airway.

In the case of incomplete (partial) filling, waste is not let down into the lower portion of the level. On the contrary, it is retained in the upper section (see Fig. 226, below) with a view to protecting the airway from excessive rock pressure, since maintenance services in this entry are rendered very difficult by the fact that the mined-out areas of the upper levels lie over its entire length (barring the levels of the top level). To keep the rocks in place under the ventilation entries, special *platforms* or waste stulls are arranged in the worked-out area, and it is on these that packing is placed.

The waste stulls should be so located as to leave ample room for all the waste lowered into the level. The waste stulls are placed on top of the props of the earlier installed support sets, which are reinforced by additional posts and one or two runs of cribs. They are extended as the production faces advance.

The fill in the gob lies at a certain angle to the line of strike. The greater the pitch of the seam the smaller is the angle. Let us imagine that the seam lies vertically and that packing material in the gob rolls down freely, without being detained either by timbering or friction with wall rocks. The angle it forms between the slope and the strike, which is a horizontal line, will be the angle of repose of the fill, that is, 38°-42°. If the angle of pitch is below 90°, the angle between the slope of the fill and the strike will be greater. It will also tend to increase because the fill is prevented from rolling freely by timbering in the worked-out area and by its friction with wall rocks. Furthermore, the initial height of the mined-out space, which equals the thickness of the seam, may, by the time packing is proceeded with, be somewhat reduced by the subsidence of the roof and the bulging of the bottom, especially in the central section of the level, and that will also delay and even stop the stowing operations. These phenomena become all the more pronounced as the coal seam grows thinner. The job of levelling the fill is assigned to special workers.

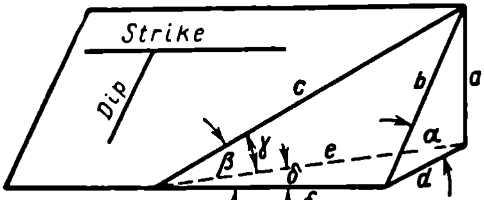



*Fig. 194.* Calculation of a fill-slope angle

Let us assume that line *c* in Fig. 194 represents the position of the fill slope. If this line runs in space at $\gamma$ angle, which is somewhat great-

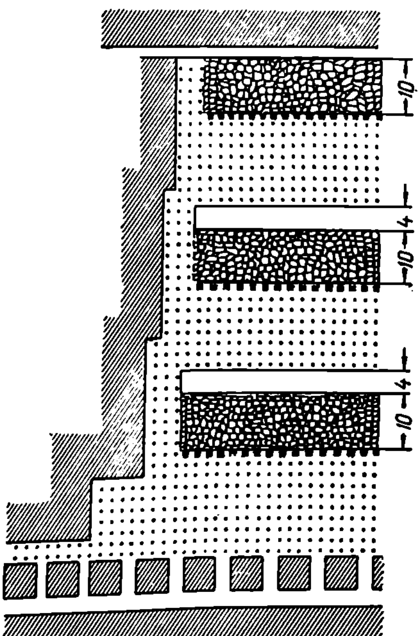



*Fig. 195.* Filling from dummy roadways (lateral entries) in steeply inclined seams

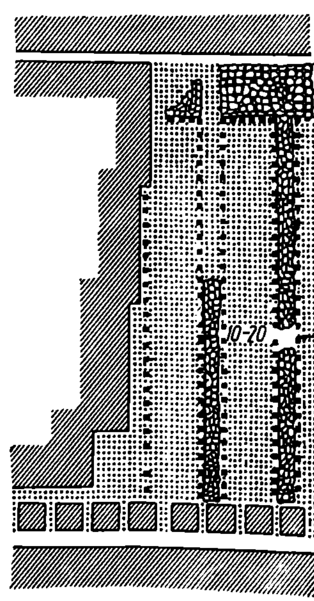



*Fig. 196.* Pack walls down the dip in mining steeply pitching seams

er than that of the fill repose, angle β, formed by the actual fill slope and strike, can be found by applying the following formula

$$\sin\beta = \frac{\sin\gamma}{\sin\alpha},$$

and the lead or advance of the lower end of the fill slope in accordance with the formula

$$f = b\cot\beta\,\frac{a\cot\beta}{\sin\alpha}.$$

It is desirable that the bench line be as parallel as possible to the fill slope.

Some research workers hold that filling material can be obtained from dummy roadways even in steeply dipping beds (Fig. 195) or pack walls be built down the dip, for which the dividing boards should be arranged along the rows of timber set props in the mined-out area (Fig. 196). Both these methods are so far hardly used.

## 16. Coal Extraction in Benches

Mining of coal in steeply inclined seams is today done by 1) air hammers; 2) coal cutters; 3) combines (cutter-loaders) and 4) coal ploughs.

1. If the seam structure permits, coal is broken off by *air hammers* (Fig. 197) without any preliminary cuts or with shallow cuts, since it is dangerous to make typical deep cuts if the pitch is heavy. Actual breaking starts with the top corner of the bench in order to protect the faceman working on the lower bench from any possible hazards.

Advanced miners have wrought radical changes in the technique and organisation of coal breaking at the faces of steeply inclined seams.

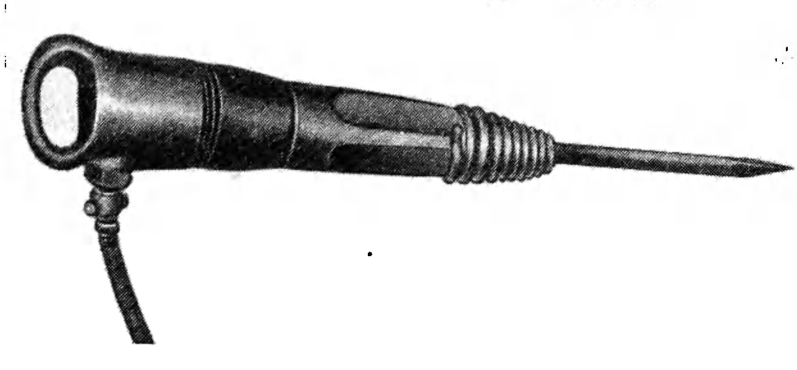


*Fig. 197.* Air hammer

A. Stakhanov, who initiated the popular emulation movement in the coal industry, set his first famous record on August 30, 1935, at the face of a steeply pitching seam in the Tsentralnaya-Irmino Mine in the Donets coal fields by breaking 102 tons of coal with an air hammer in 6 hours.

Prior to the introduction of the differentiated method of work in breaking coal with air hammers, mining in overhead stopes followed the hand-breaking pattern, that is, each bench was worked by one man who both broke coal and timbered the face.

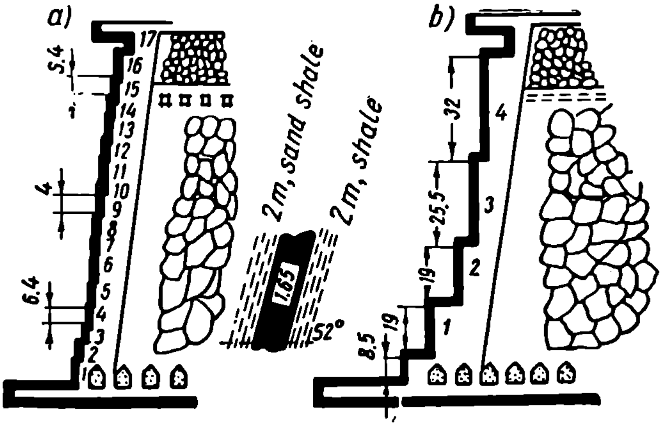



*Fig. 198.* A face at the Mazurka seam
*a*—prior to adopting a differentiated method of work; *b*—after adopting it

Stakhanov suggested a novel way of organising mining at the faces of steep beds, assigning the breaking of coal and timbering to different men. This made it possible to raise substantially the efficiency of facemen and make a better use of the pneumatic hammer.

The new method of mining sharply increased per capita efficiency at the face, 1.5-3 times and over on the average, this being a fact of extreme importance. The length of the benches increased to 20, 30, 40 and more metres, depending on the firmness of coal and the thickness of the bed (Fig. 198).

High efficiency, of course, is achieved not only by an increase in the height of coal benches and differentiation of labour, but, in this instance, also by intensive and skilful work and by practical application of the principles enumerated in Section 14.

High benches have a number of other advantages.

The most difficult operation in working benches is the cutting out of "corners". In the case of high benches, their number in the level is proportionally smaller, and this also reduces the total number of corners.

To make better use of hammer weight, coal in the bench is broken downward. Each long bench is serviced by one faceman and one, two and sometimes more timbermen. The latter lag somewhat behind the faceman (Fig. 199). To protect the faceman from falling timber *safety platform A* is installed on support props over him.

Hammers are most effective in breaking coal of medium hardness, or hard but fissured coal. It is of prime importance that air pressure at the face should be kept at not less than 4-5 atm, for if it drops below that it would lead to a rapid decrease of air hammer efficiency. In good condition, the hammer consumes from 28 to 40 cu m of free air per hour of net work, depending on its size; air consumption increases as it wears out. The depreciation rate for an air hammer is 2,000-2,500 hours of continuous operation.

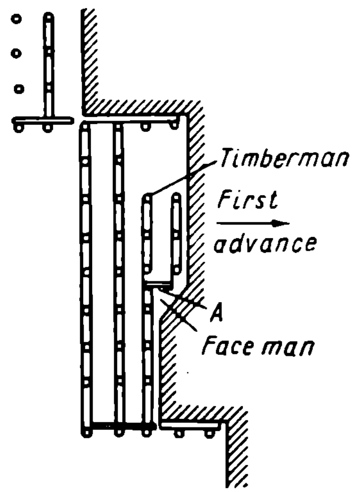



*Fig. 199.* Sequence of coal breaking and timbering in high benches

The air hammer is better utilised in mining with high benches, since the duration of its actual work during the shift is then much greater. At the same time, the number of hammers engaged in the mine section drops sharply, this facilitating the laying of air lines, cutting down the number of hoses, tee-pieces, cocks, etc., and reducing leakage of air, its consumption, and the quantity of lubricants used.

In the Ruhr coal fields (West Germany), where sloping and steep seams predominate and coal and wall rocks are relatively soft, coal breaking with pneumatic hammers is the principal mode of stoping.

The greatest disadvantage of pneumatic hammers is the high cost of compressed air, whose production consumes much electric power because of the low efficiency factor of pneumatic plants and tools. Hence the efforts to design an efficient *electric* coal hammer. However, the big and persevering efforts put in by designers (N. Shmargunov, L. Grigoryev, N. Komarov and others) to this end have failed to produce a coal hammer suitable for field work.

2. In conditions prevailing in steeply inclined seams *coal cutters* are efficient only in individual instances.

The production face (Fig. 200), in this case, should be straight and run on the dip. Its lower portion should have some lead so as to allow the coal-cutter bar to penetrate under coal and provide enough room for storing broken coal. When undercutting, the machine moves uphill, pulled by a feed rope. For safety's sake, there is yet another rope, the lower end of which is fastened to the cutter and the upper wound around the drum of a special hoist set up in the upper entry. The safety rope is kept taut at all times so as to prevent the machine from falling if the feed rope breaks. The machine is lowered on an idle run by the safety rope. Soft coal breaks off by itself in the process of undercutting. Hard coal is blasted by explosive charges in drill holes. To facilitate the breaking operation, curved cutter bars, which make cuts of singular shape (*b*), can be used instead of the ordinary straight ones (Fig. 200, *a*). The method used in face timbering is shown in the figure.

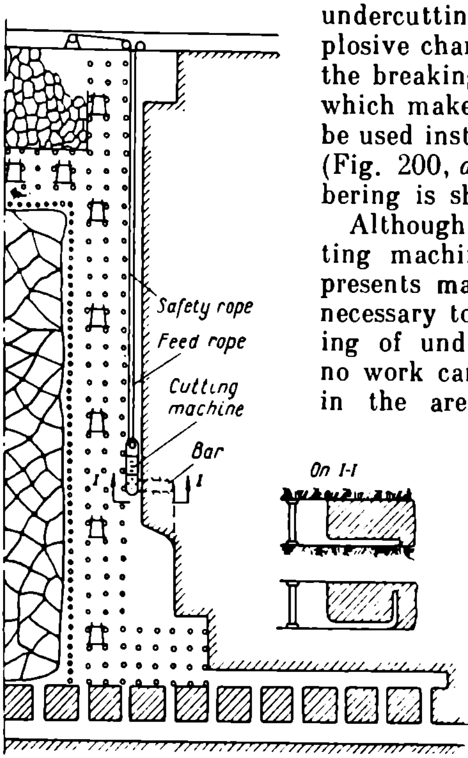



*Fig. 200.* Mining of a steeply inclined seam with a coal-cutting machine

Although possible, operation of coal-cutting machines in steeply pitching beds presents many technical difficulties: it is necessary to maintain a safety hoist, breaking of undercut coal is inconvenient, and no work can be done in a straight dip face in the area below the coal cutter. The result is that the efficiency of these machines is far inferior to that recorded in gently sloping seams. The worse the wall rocks, the more inconvenient it is to operate the coal cutter. It is particularly so in the case of broken beddings. Because of that the use of coal-cutting machines in steeply inclined seams is restricted to exceptional cases, for instance, when there is a combination of a regular seam occurring in firm country rocks and coal that is very hard but readily detachable from the back.

3. The ККП-1 coal combine (combine for steep seams, Model 1) was introduced for mining steeply pitching beds in 1950. The first pilot series of these machines for mining steeply inclined seams 0.8-1.5 metres thick, containing medium-soft coal and occurring regularly in stable wall rocks at an angle of 50°-75°, was put out in the following year.

This machine (Fig. 201) was invented by engineers A. Zasyadko, A. Topchiev, V. Balykov and A. Pichugin. It operates downhill,

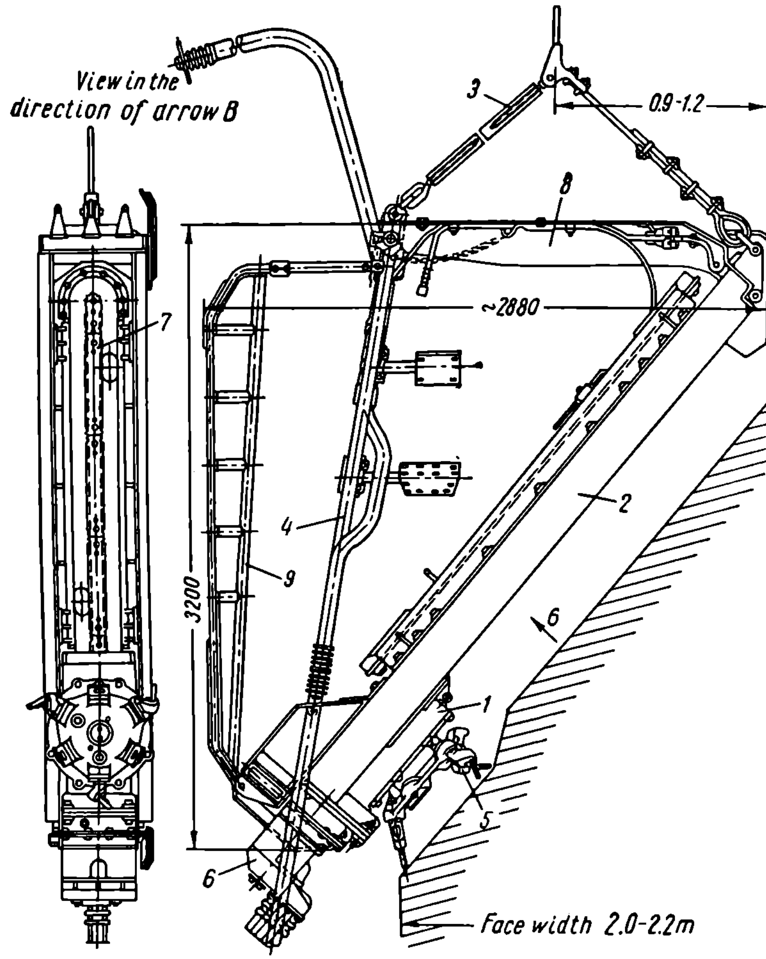



*Fig. 201.* Diagrammatic representation of a KKII coal combine

along the dip of a seam, and draws coal in a diagonal bench, separating a band 2-2.2 metres wide. The combine's bed frame *2* (Fig. 201) accommodates reciprocating breaking-down or ripping machine *1*. It consists of three-point rip bit *5*, compressed air motor *6* with a rated capacity of 32 hp and a reduction gear. The bit rotates at a rate of 50-60 rpm. The ripping machine moves along rack *7* and the bed frame over a distance of 2.2 metres at a speed of 17 m/min.

Side frame *4*, hinged to bed frame *2*, and safety platform or cover *8* constitute the main body of the combine, which also has suspension device *3* that can be attached to the rope of the hoist set up in the upper entry.

As the bit, performing simultaneously rotating and reciprocating movements along the bed frame, separates coal in strips 100-200 mm thick and it slides down the sloping bench, the machine is fed downhill at the rate of 0.4-0.5 m/min. The hoist is started and stopped by remote control, the operator being all this time on the combine under the protection of steel cover *8*.

To prevent the combine from leaving the contours of the face, there is lateral slide *9* pressed against the first row of face timbering. Coal in the seam is broken, depending on its hardness, by the bit rotating in one or both directions. In the latter case the machine's output may be brought to 50 tons per hour (in a bed 1-1.3 metres thick).

The space freed by the extraction of coal bands has to be supported by timber sets, with 0.9-metre intervals between rows. The back of the seam should be lagged with slabs. The only unsupported thing is a strip about a metre wide near the breast of the face, along which the combine is brought to the upper portion of the coal wall after the mining operations are over. When a combine moves ahead at a rate of 0.4-0.5 m/min, two or three timbermen do not always manage to put up the supporting sets in the area near the face, and this causes interruption in its operation. To remedy this, the designing organisations are now busy elaborating *a mechanised or powered metal support* which would permit continuous operation of the combine in absolute safety.

When the ККП coal combine was first introduced at the Surtaikha Mine, inventor Babarykin suggested lowering a special suspended metal stage or platform right after the combine to facilitate and accelerate face timbering. The platform accommodates a timberman and a stock of mine timber. It is lowered and hoisted (folded because its parts are hinged) by a rope from a hoist set up in the upper entry, and is operated by a timberman with the aid of push buttons.

After the extraction of a coal strip, the combine stops on the bottom 6-metre bench, the lateral slide is taken off, along with the bit holders, and the frames are folded prior to its hoisting to the upper portion of the wall by rope. The lifting power of the hoist is 4 tons, the drum diameter—600 mm, and it accommodates 240 metres of 21-mm rope. The combine is hoisted at the flitting or tramming speed of 4-5.7 m/min. All operations connected with the dismantling, transfer and preparation of the machine for the next cycle are performed during the back shift.

The combine and the hoist may be powered both by electric (17 and 11 kw) and compressed-air (30 and 16 hp) motors. The overall dimensions of the combine in the operating position are: length (on the dip) —3 metres, width—2.9 metres; height, depending upon the thickness of the bed—0.76-1.04 metres; weight—3 tons.

In the average conditions of steeply pitching seams in the Donets coal fields ККП combines are capable of producing between 9,000 and 11.000 tons of coal per wall per month.

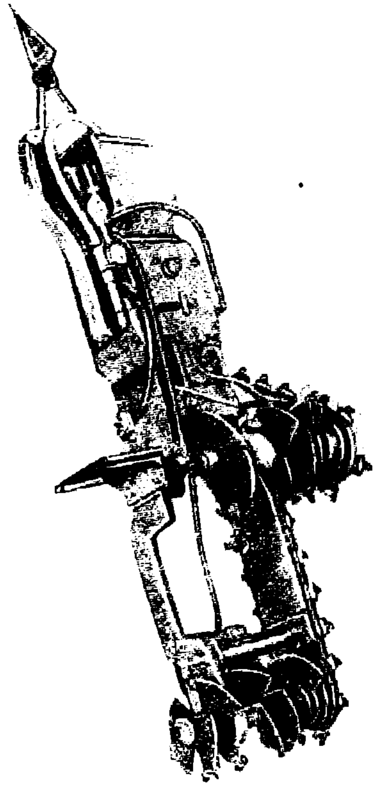


*Fig. 202.* K-32 coal combine

The Gorlovka works has put out a pilot lot of УКШ-1 combines (screw conveyer combine, Model 1), designed by N. Ignatov for working steep seams 0.7-1.1 metres thick in coal walls 70-150 metres long with soft and medium-hard coal and firm and medium-stable wall rocks. The operating mechanisms cutting coal are the chain bars and screws with cutting bits and shearing plates. Screw conveyers also remove gum from the face. The combine is serviced at the face by six men (machine runner, his helper, electrician, hoistman and two timbermen).

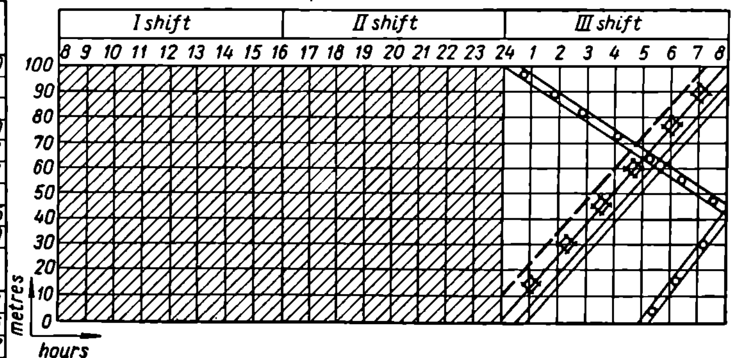
During the tests in a coal wall 82 metres long the machine completed one cycle in eight hours.

Field tests of the K-32 type coal combine (Fig. 202), also designed for mining of steeply inclined seams, are at present under way. The operating mechanisms in this machine are the screw conveyers fitted with bits (teeth) and ripping wedges. The rated capacity of the electric motor actuating the combine is 30 hp, the weight of the machine—3.7 tons and its range—1.05 metres.

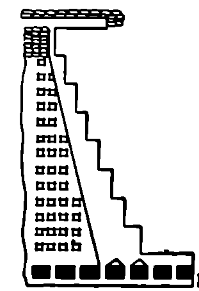
4. Mining in steep seams can also be done by coal planers. Technically, this method of mining in the conditions prevailing in steeply inclined beds includes a number of features differing from those of slightly sloping seams: the conveyer is absent; broken coal descends

| Nos | Technical and economic indices | Unit of meas | Value |
|---|---|---|---|
| 1 | Wall length | m | 100 |
| 2 | Seam thickness | m | 1 |
| 3 | Pitch angle of the seam | degree | 45 |
| 4 | Timber width | m | 0.9 |
| 5 | Advance per round | m | 1.8 |
| 6 | Output per 1 m² of seam area | t | 1.3 |
| 7 | Operation cycles per day | – | 1 |
| 8 | Coal output per operation cycle | t | 235 |
| 9 | Daily production program | t | 210 |
| 10 | Roof control | Partial filling and roof subsidence | |
| 11 | Labour actually engaged per day | – | 32 |
| 12 | Planned output per actual man/shift | t | 6.6 |
| 13 | Total number of mine workers | men | 4.2 |
| 14 | Planned monthly output per man in the wall | t | 150 |

Operation chart

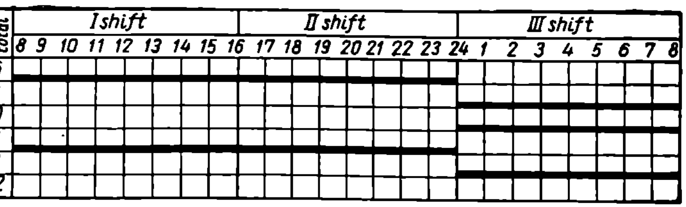



Face at the beginning of the I shift



Labour distribution chart

| Nos | Classes of work | Shifts I | Shifts II | Shifts III | Grand total |
|---|---|---|---|---|---|
| 1 | Face men | 8 | 8 | – | 16 |
| 2 | Air-line service men | – | – | 2 | 2 |
| 3 | Timber supply and crib men | – | – | 10 | 10 |
| 4 | Chute pullers | 1 | 1 | – | 2 |
| 5 | Loading chute transfer men | – | – | 2 | 2 |
| | Total | 9 | 9 | 14 | 32 |



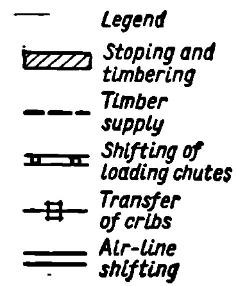




*Fig. 203.* Cyclic operation schedule for a steeply pitching seam

by gravity; when running idle the planer can move downhill by its own weight.

Field trials with a coal plough in mining Pugachovka coal seam of the Donets basin, 0.8 metre thick and dipping at 60°, were carried out in 1949. The tested planer was the one designed by Y. Nekrasovsky. It was made of a solid metal frame placed on skids and fitted with cutting teeth. Its weight was 1.7 tons. The face was straight, but slightly inclined, so that its top corner was somewhat in advance of the bottom one. The coal plough cut coal as it was pulled along the face by a rope. When it ran loaded uphill, the rope wound around the drum of a hoist set up in a cross heading parallel to the haulageway. Adjustable idler pulleys were provided at the corners of the face to guide the rope. The rope was also passed over special rollers on the coal plough in order to press it against the breast of the face. The latter was supported by props and cribs. Timbering was put up after the face had been advanced by about 0.8-1 metre. When the plough moved downhill by its own weight, the electric motor of the hoist was shut off, and the speed of the descent was regulated by the brakes of the hoist. There were no men present at the face during the operation of the plough.

The field tests of the coal plough proved successful. Labour efficiency and coal tonnages increases were greater than those in the case of air hammers. It is quite possible that the employment of coal plough in steeply pitching beds will make it possible to work very low seams.

## 17. Work in a Steep Face

Work in coal walls of steeply inclined seams should be organised on the basis of a cyclic operation schedule. The main provisions relating to the cyclic organisation of stoping operations, formulated before, fully apply to the conditions prevailing in steep beds.

Fig. 203 is illustrative of a cyclic operation schedule drawn up for one 24-hour cycle for the overhand stope of a steep seam worked by air hammers.

CHAPTER XII

# CONTINUOUS METHODS OF MINING

## A. GENTLY PITCHING AND MEDIUM-STEEP SEAMS

### 1. Development Work

In Chapter II we have seen that the levels of gently sloping seams are opened up through permanent inclines and slopes or directly through inclined shafts, from which level drifts or entries are driven.

Fig. 204, for example, shows main hoisting shaft *A* with workings adjacent to it, air shaft *B* and break-through *1-2* between them. The break-through is made mostly in the shape of two parallel openings connected by *cross headings*. This facilitates ventilation during the drivage of the break-through. The completion of the break-through connecting the main hoisting and air shafts makes available two *independent exits* to the ground surface, and this allows normal ventilation of underground workings. One of the shafts is for the *downcast current* of fresh air and the other for the *upcast return current*. No underground stoping is permitted before the two shafts are properly connected and normal ventilation is ensured.

If there is more than one level in the up-dip portion of the mine field, that is, in the section lying above the shaft station (in Fig. 204 there are two such levels), the break-through is used for arranging *permanent slope 1-3* to allow conveying the coal drawn in the upper level to the shaft.

Lower haulageway *3-4* and upper ventilation entry *2-5* are run starting from the break-through level drifts. The distance between these two openings is equal to inclined level interval $h$. The entries on the other side of the level are pushed forward in analogous fashion. Entries running from the permanent incline in the mine field along the dip are driven in the same manner.

The entries are timbered by supporting sets of different materials and design as conditions require it.

The materials that can be used include timber, metal, concrete and reinforced concrete. The sets themselves may be made of straight or bent members, complete or incomplete, rigid or yielding.

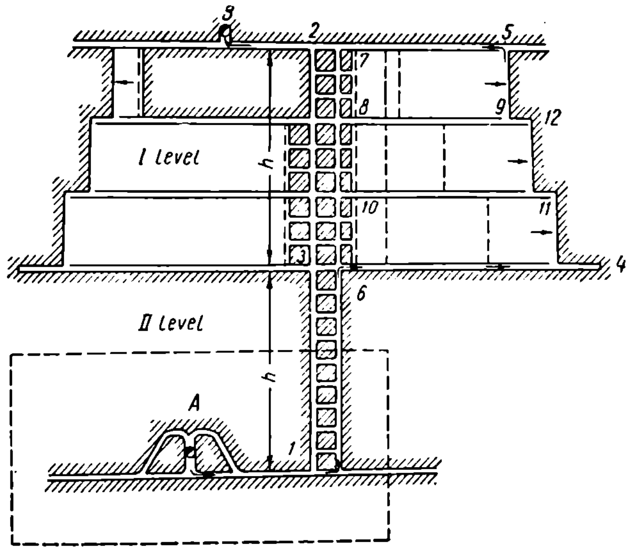



*Fig. 204.* An example showing the opening up of a deposit and development of levels

Timber sets of trapezoid cross-section (Fig. 205) can, when necessary, be reinforced with roof timbers (rafters) (Fig. 206). If reinforced concrete pipes are employed as posts (Fig. 207), the head pieces, to resist rock pressure, should be made of steel I-beams with saddles covering the upper ends of the posts. The shape of metal supports may be trapezoidal (Fig. 208), arched (Fig. 209), horseshoelike (see Figs 221 and 222), and, in the case of unfirm bottom rocks, even ring-shaped (Fig. 210). Figs 221 and 222 illustrate timbering of entries in steeply pitching beds, but such supports can also be used in slightly inclined and sloping seams.

Figs 205 and 208 depict dimensions of workings and the size of support members used.

When entries are subjected to strong rock pressure, particularly if they are in the vicinity of mined-out areas, the support should be made *yielding* or *pliable*, allowing the workings to change their dimensions and shape within a certain range. Rigid support, for instance of the types depicted in Figs 207, 209 and 210, is permissible only when rock pressure is insignificant—if the opening is a lateral drift or protected by safety pillars of considerable size, or else when the rocks over the worked-out spaces have subsided completely, that is, when high rock pressure caused by the caving and settling of roof rocks has ceased to produce effect. Mine timber of

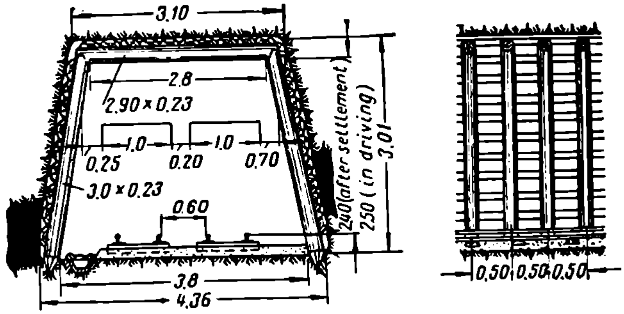



*Fig. 205.* Entry timber support

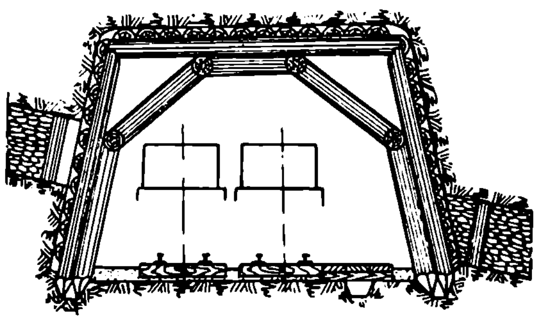


*Fig. 206.* Entry timber support with rafter reinforcement

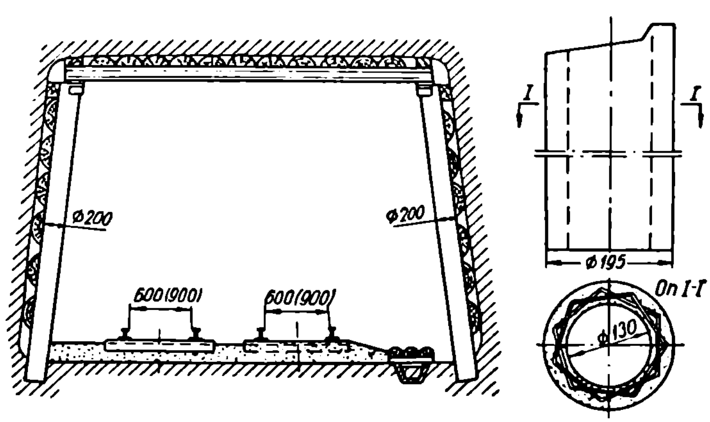



*Fig. 207.* Reinforced concrete support posts in an entry

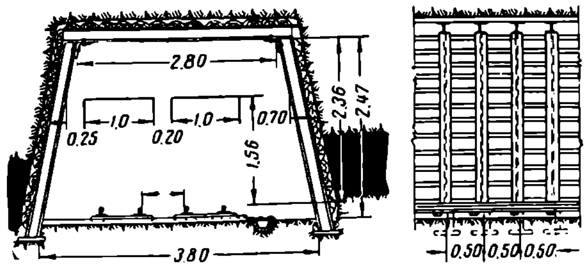



*Fig. 208.* Trapezoid metal support

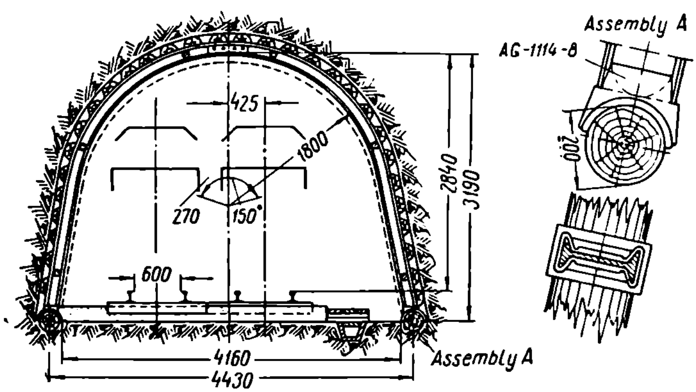



*Fig. 209.* Bent metal support

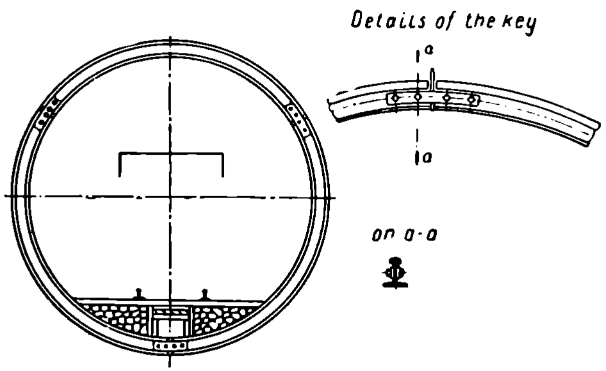



*Fig. 210.* Ring metal support

ordinary design is capable of yielding somewhat under pressure, but to make this feature more pronounced the lower ends of the props should be tapered (burring props), as shown in Fig. 205.

To make a metal support more pliable, its members are apt to develop a certain play (see Fig. 222) under heavy rock pressure.

The entries are protected from the mined-out areas by coal *pillars* or *pack walls*, that is, by filling materials. Permanent slopes, inclines and their manways are also protected by coal pillars (see Fig. 204) or, less frequently—in low seams, by pack walls. In order to block out such pillars, *crosscuts 6-7* (see Fig. 204) are made to run parallel to the break-through. As they advance, these crosscuts are connected with the break-throughs by *cross headings*.

*Stoping operations* are started from crosscuts.

## 2. Longwall Mining

If there is just one continuous straight dip face (*wall*) in the levell this modification of continuous mining is designated as longwal, workings. For example, Fig. 211 is illustrative of the longwall mining with a coal combine of a low slightly inclined seam. Roof control involves complete caving. The lower and top entries are protected by pack walls. The flight conveyer at the face loads coal directly into the mine cars of a train driven by an electric locomotive.

In this system of mining, production faces are generally distinguished by their considerable extent. Long continuous faces possess a number of major advantages:

1. They yield more coal.

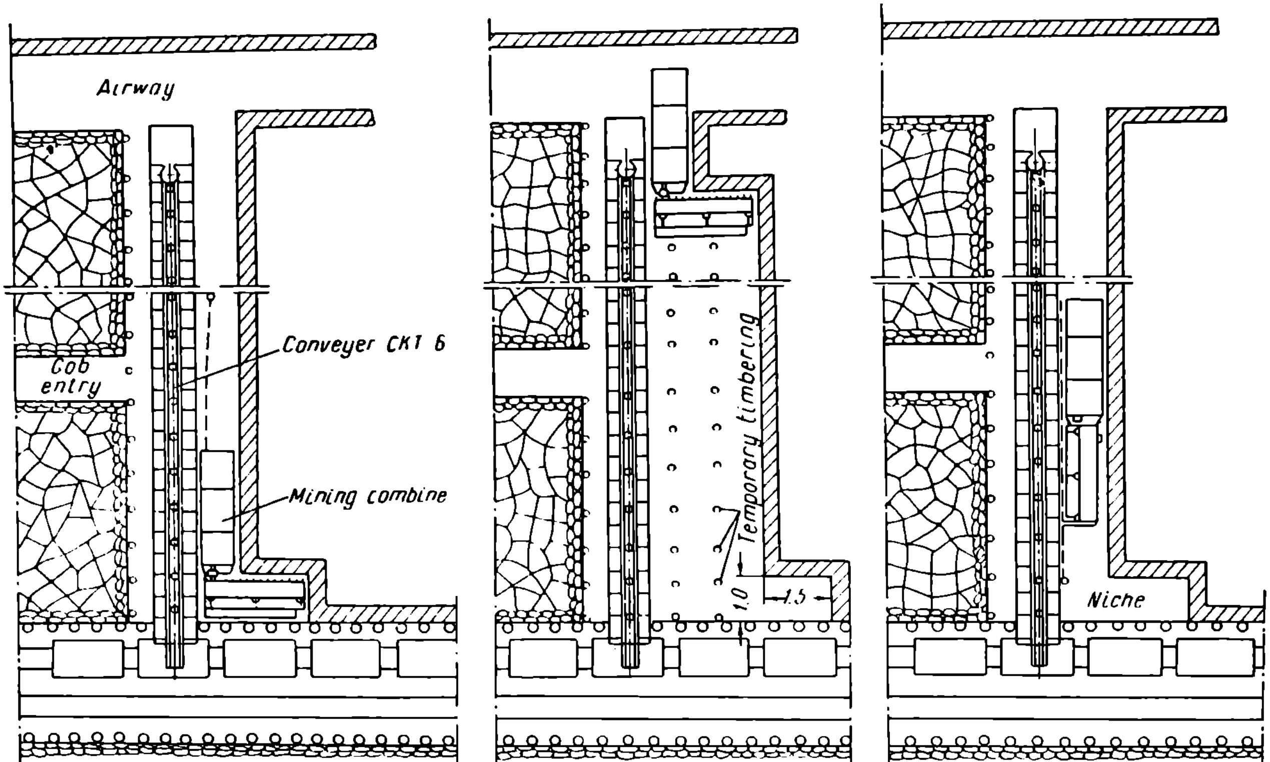
2. They allow the maximum use of coal combines, cutting machines and conveyers. When necessary, longwalls can be serviced not by one, but by two combines or coal cutters. A face extending for 100 metres can be serviced by one conveyer. In walls of greater length two or three conveyers can be put up in tandem.

3. In longwalls, all other conditions being equal, the production programme can be fulfilled with a minimum number of active walls. This reduces the number of development openings and, consequently, the cost of their maintenance, and simplifies the transportation system.

4. Supervision of mining operations is much simpler.

On the other hand, longwall mining may cause a series of inconveniences:

1. Inasmuch as a coal wall of appreciable extent is a major production unit at the mine, any delays in its operation may seriously affect the output of the mine as a whole. Hence the need to keep active longwalls in perfect order and, besides, to make provisions for stand-by or reserve faces (Section 9).




Starting of the combine operation at the face | Completion of the combine operation at the face | Lowering of the combine in the wall

Fig. 211. Longwall mining

2. One of the causes leading to hitches in the normal operation of longwalls is the availability of geological disturbances: faults, pinches or squeezes of the seam, abrupt changes in the angle of dip. Therefore, continuous faces of considerable length can be worked in regular and uniform beds only. Gradual, limited changes in the angle of dip are no obstacle to longwall mining.

3. A disadvantage of longwall mining is the varying properties of wall rocks, for it makes it necessary to change roof-control and face-timbering methods, and to resort to reinforced support in some places.

4. In faces of great length there may be difficulties with ventilation. The longer the face the greater, generally speaking, should be the volume of fresh air supplied for its ventilation, for there are more men engaged in it, more explosives consumed (if they are used), and more methane evolved (if the seam is gassy). Inasmuch as the cross-section area of the space near the face ordinarily remains constant along the entire length of the wall, the velocity of the ventilation current increases proportionally to the volume of air reaching the face. According to safety regulations, however, this velocity should not exceed 4 m/sec. Consequently, ventilation conditions may become a factor restricting the length of a continuous face.

5. When, because of its length, a coal wall is serviced by two or three conveyers set up in tandem but actuated by separate drives, a stoppage of the lower one leads to the stoppage of the upper ones, and this disrupts work in the whole of the wall.

6. Passage of men and delivery of mine timber are made all the more difficult by the length of the coal walls, especially in low seams.

Hence the use of longwalls of considerable extent requires: 1) regular occurrence of the bed and uniform properties of the enclosing rocks; 2) concurrence of the face length and permissible velocity of air current; 3) equipment of the coal wall with mine machines of sufficient capacity; 4) systematic roof control; 5) well-elaborated organisation of work in accordance with the cyclic operation schedule; 6) availability of stand-by or reserve longwalls (Section 9).

When all these conditions are complied with, the advantages of the longwalls described above become so important that in the prevailing geological and mining conditions the chosen extent should be as great as possible. For this reason it is often with much success that longwalls of considerable extent (100-150 metres) and even more (150-300 metres) are used in mining practice.

### 3. Continuous Sublevel Mining

Since in mining gently sloping beds the inclined level interval may be as high as several hundred metres, it is not always feasible to practise the longwall method of mining as such, and one has then to resort to the more complex system involving the *division of the level* into two or, though less frequently, into a greater number of *sublevels* (see Fig. 204).

Thus, in addition to level drifts *3-4* and *2-5*, so-called sublevel entries *8-9* and *10-11* are driven within the boundaries of the level. These are also called *intermediate entries*. The distance between intermediate entries, measured on the line of dip, is called *sublevel interval.*

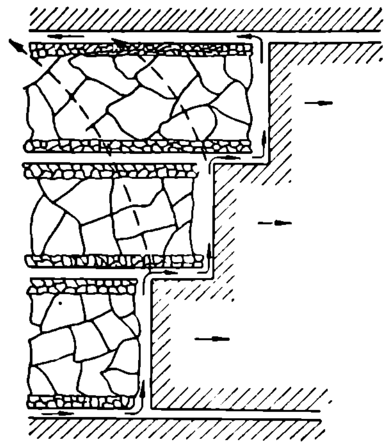


*Fig. 212.* Face advance in upper sublevels

*A sublevel*, consequently, *is a portion or section of a level lying between two neighbouring intermediate entries or between the main entry and the intermediate level adjacent to it.* The latter naturally refers to the uppermost or lowest sublevels of a given level.

When the continuous method of mining is employed, a single longwall is usually started in each sublevel. Therefore, in this case the sublevel interval, that is, the extent of the longwall, should also be as great as possible in the existing conditions.

Within the range of each sublevel coal is mined in the corresponding independent longwalls. The distance between individual working faces *9-12* is designated as *face advance* (see Fig. 204). Fig. 204 is illustrative of an instance when it is the longwall of the *lower* sublevel that *goes in advance*, while Fig. 212 shows the face of the *top* sublevel *advancing first.*

When a level is divided into sublevels, a *slope* has to be arranged within the boundaries of the level to allow the transportation of coal from working faces. The longwalls of each sublevel are connected with the slope by independent openings.

Intermediate entries are provided with mine tracks to haul coal in mine cars or else with conveyers. For example, coal extracted in working face *5-9* (see Fig. 204) of the top sublevel is loaded into mine cars (or onto a conveyer) at point *9* and enters the slope at point *8*. When work proceeds according to the pattern given in Fig. 204, stoping starts in the lowest sublevel as soon as a crosscut is made in it. Simultaneously, a coal heading is driven in the second sublevel. The further advance of production faces will be on strike. An analogous picture will be seen on the other side or flank of this level.

Hence *no development work is done in the solid mass of the mineral in front or ahead of longwalls.* This is the main distinguishing feature of the method of continuous mining. But there is an important reservation to be made in this connection: all we have said above does not apply to the main entry, whose heading should be driven considerably (by 70-100 metres) in advance of the production face (that is, it should be in the position shown, for example, in Fig. 212). This is necessary to make loading of coal delivered from the longwall completely independent from the drivage of the entry and to facilitate train switching at the loading station of the longwall.

A high advance rate of the main entry heading is also of importance for exploration and study of the details of the occurrence of the seam and, if necessary, for the development of new production places (longwalls).

The arrangement of the sublevels depicted in Fig. 204 is characteristic of the fact that the faces of the lower sublevel are ahead. The reverse disposition of working places is depicted in Fig. 212.

The following are the advantages gained in the first case. The ventilating air current, whose direction in Fig. 204 is indicated by arrows, sweeps the working places when ascending and thus contributes to the better ventilation of stoping areas than can be achieved in the second instance, where it may leak through the mined-out space, seeking to reach the air shaft by a short cut (dash arrows in Fig. 212).

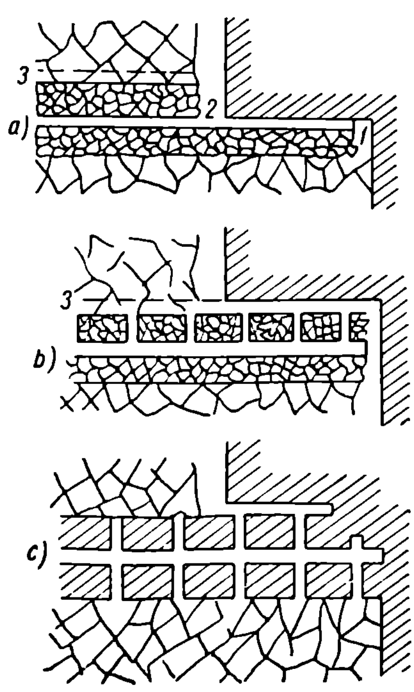



*Fig. 213.* Intermediary entry protection by rib fills or chain coal pillars

When water appears in mine workings, it flows down the maximum gradient, that is, to the dip, and it is for this reason that, in the first instance, it runs from any sublevel directly into the mined-out area, by-passing the working places of the underlying sublevels, and, in the second, it gets to the longwalls below.

In continuous mining, the entries are protected from the deleterious effect of rocks caving in over mined-out spaces by *rib-fills* (pack walls) of wall rocks blasted during their drivage, built near these openings, or by safety rib or chain pillars of coal left on both sides of them (Fig. 213). This last method, however, is to be avoided, for it is fraught with many inconveniences for the operation of mine machines at the face, requires a great deal of additional development openings and involves considerable losses of coal in safety pillars. But in the case of unfirm rocks and medium-thickness seams, haulageways sometimes have to be protected by safety pillars. This method hampers the extraction of coal from a longwall, for it requires

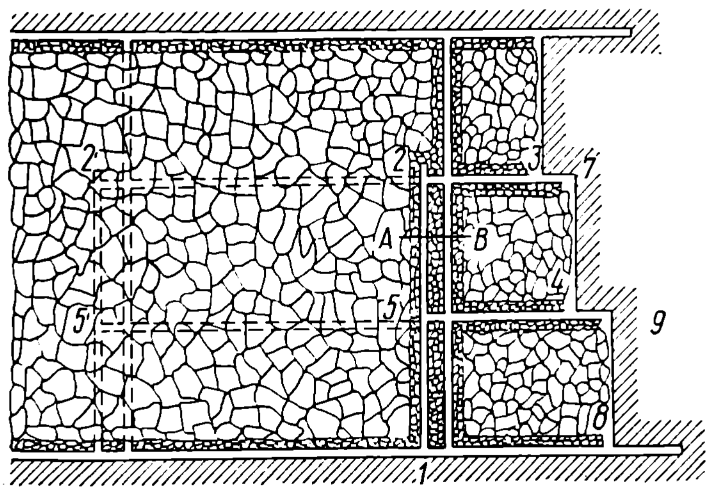



*Fig. 214.* Intermediary entries and slope in goaf

two short supplementary conveyers to be put up along the crosscut and coal heading.

The most favourable scheme for the transportation of coal is *a* in Fig. 213. But in order to avoid tramming blasted rock from the entry face and hoisting it up the rise at point *2*, it is sometimes preferable to put the entire amount of blasted rock into the lower oblique heading right on the spot and to obtain the fill to be stowed over the main entry by slashing gateway *3*, which is specially made for this purpose in the worked-out area.

As the production faces continue to move away from the initial point in the flank of the level, the length of the intermediary entries increases and so do the costs of transportation and maintenance of these openings. Therefore, beginning from a certain distance, it is better to cease using intermediary entries for coal haulage and to discontinue their maintenance. To make this possible slope *1-2* is made in the mined-out area (Fig. 214) and then coal is hauled from production places along the intermediary entries only over distance *2-3* and *4-5* to the slope and then through the slope to main gangway *1-6*. Sections *2-2'* and *5-5'* of the intermediary entries then become useless and are abandoned. If there are rib pillars of coal near them, it is recovered as far as possible. The tracks are torn off and taken away. Whenever possible, some of the timber supports are knocked down.

Down the slope coal can be transported in mine cars by rope-haulage. But that can be done better by conveyers. This is especially desirable when conveyers are used in the intermediary entries, too.

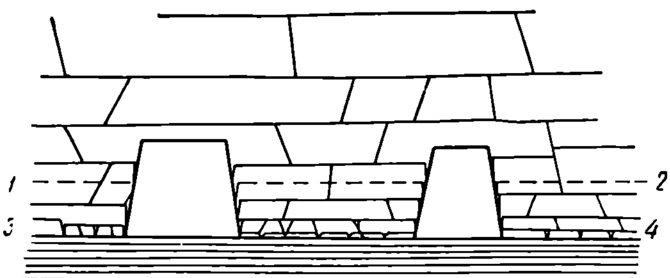



*Fig. 215.* Making a mine slope and manway in goaf

When the slope is serviced by a conveyer, the passageway for men is provided in the slope itself, side by side with the conveyer. Mine tracks for the delivery of timber and equipment to working places and for the tramming of barren rock, etc., are laid in a separate opening parallel to the slope. They are laid on the side of the slope opposite the working face. In slopes equipped with belt-and-flight conveyers, tracks can be laid in the slope itself, alongside the conveyer, since timber supplies and, if necessary, waste rock can be transported by the conveyer, and mine tracks are not often used—only for the transportation of heavy objects. At this time men's passage in the slope should be stopped.

The slope and its manway are driven in a more or less caved goaf (see Fig. 215 which gives section *A-B* of Fig. 214 on a larger scale), where back *1-2* has already subsided into position *3-4*, in places touching the bottom. That is why the removal of blasted rock from openings made in goafs to mined-out areas presents many difficulties and is sometimes altogether impossible. This involves transporting considerable amounts of barren rock from the headings. Besides, the country rocks in which openings are driven quite often are seriously disturbed by previous cavings and subsidences. All this makes driving of slopes and manways in goafs a costly and slow operation.

When working faces move away to a certain distance from the existing slope, a new one is arranged. To distinguish it from a permanent slope, it is called *intermediary. The part of a level mined through any such intermediary slope is called mine block.* Ordinarily slopes are numbered in the order of their establishment and denoted by cardinal points (for example, Western I, Eastern III, etc.). A portion of the mine block lying between two neighbouring entries and slopes (for instance, *2'-2-5-5'*) is sometimes called *working section.* This term should not be confused with *mine section*, which is a part of the mine field in charge of an engineer (or technician)—the mine section superintendent.

The higher the cost of making and equipping a slope the greater the distance between intermediary slopes; and the smaller it is, the higher the maintenance and haulage cost for each one of them. The distance is usually 150-200 metres for the one-way slopes so far discussed.

Basic knowledge of the way the distance between intermediary slopes is calculated is presented below (Chapter XIII).

Coal in intermediary slopes is transported by conveyers (belt-and-flight).

Arrangement of slopes in mined-out areas has a number of major disadvantages:

1. As already stated, their driving in goafs is a costly and a relatively slow operation.

2. Openings run all the way amid caving zones and their upkeep is therefore a costly affair—at least as long as the rocks of the back do not come down to the bottom and caving and subsidence near the openings do not cease altogether, a thing which generally occurs after a considerable lapse of time.

3. Slopes in goafs can be started only when working face *8-9* of the bottom sublevel (Fig. 214) has advanced at least scores of metres from initial slope-site *2-1* and the manway, so that their sinking will not interfere with the operations going on in entry *6-8*. Since the working faces continue to move forward during the sinking of a slope and manway, when slope *1-2* becomes operative throughout its length, intermediary entries *2-7* and *5-9* will have advanced a considerable distance. For this reason engineer A. Belinsky suggested driving a long slope simultaneously by headings started from several dummy roadways. This was done in 1937, at Shcheglovka Mine No. 1 in the Donets coal fields. The slope was run along the face of a 180-metre coal wall in a seam 1.3 metres thick, involving the slashing of wall rocks. Its size was: width at the bottom—4 metres, at the top—2.85 metres, height—2.5 metres. Mining in the coal

wall was stopped for the time. The slope was driven in a direction determined in advance by the mine surveyor simultaneously through seventeen raise slopes, each starting from a separate dummy roadway. The waste from slabbing was stowed away in dummy roadways. The job was done by 142 men divided into 17 teams. In spite of its large section the 180-metre slope was completed in two days.

Because of the disadvantages of this arrangement of slopes in goafs, continuous mining according to this modification has been practised but very rarely in the past fifteen years. Today, however, with the tendency towards greater level intervals, it is likely to be employed more.

Use can be made of yet another method, the one of driving intermediary entries in an *intact* solid mass of coal. To accomplish this, the entries—haulage and ventilation—are pushed forward far ahead of the production faces (Fig. 216).

This system implies driving a series of development openings in front of the working faces and the method can no longer be referred to as one of continuous mining, but as a transitory one from continuous to pillar systems. Its basic merits are: 1) slopes and manways do not run in goafs, but in intact solid masses of coal, which serve as safety pillars; 2) seam occurrences are thoroughly explored through development openings run ahead of production faces.

The system also has its shortcomings: 1) openings driven in a solid coal mass require a more complex ventilation system, especially in gassy mines; 2) driving these openings ahead of time demands considerable outlays. Nonetheless, the method is used quite frequently.

In continuous mining, *ventilation* of development headings and working faces is distinguished by simplicity. The pattern of air movement is simplest in the case of the longwall variety of the system. With sublevels a simple ventilation scheme is achieved when the same air current consecutively sweeps all the coal walls of the level. But in this instance the upper walls may be swept by an air current containing methane. For this reason any possible use of successive (through) ventilation of coal walls with a single air current is subject to the following conditions laid down in the Safety Rules:

"Each working face and the headings of the adjacent development openings should, as a rule, be ventilated by a separate current of fresh air.

"Consecutive ventilation of several simultaneously operated coal walls (production faces) is permissible in mines working seams with no hazards of sudden outrushes and blowers and only in following conditions:

"a) the spacing on strike between the adjacent coal walls must not exceed 20 metres in the case of mining low seams and 40 metres in medium-thickness

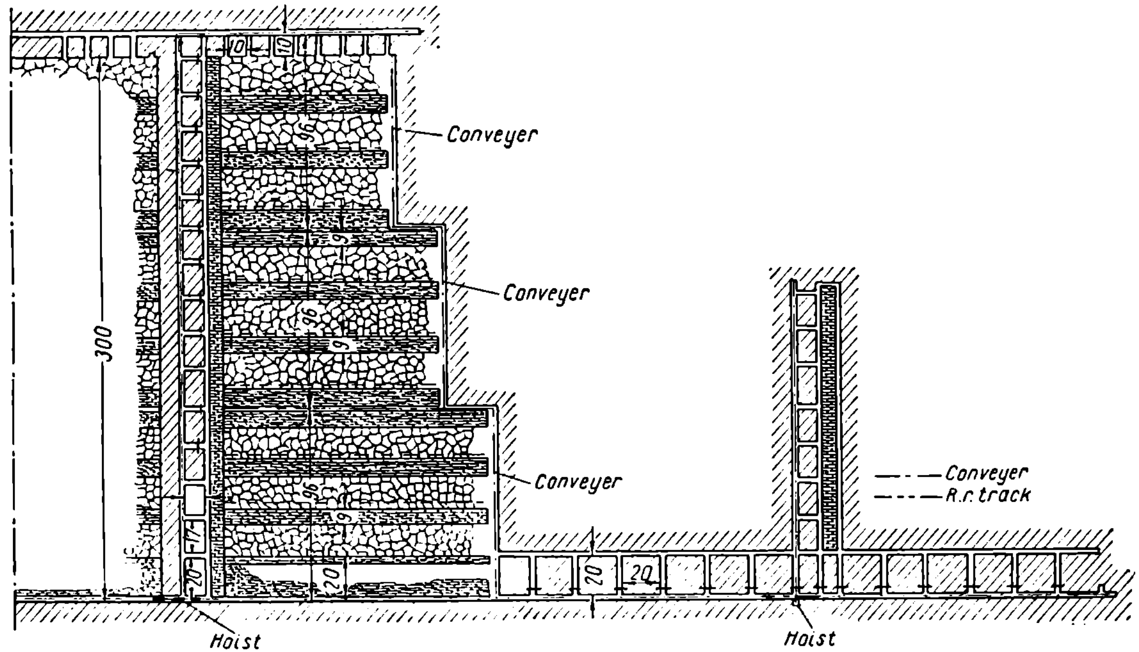



*Fig. 216.* Continuous mining with a mine slope driven ahead of production faces

and high seams. In this instance successive ventilation of coal walls connected up the dip by a ventilating break-through is inadmissible.

"In nongassy beds and in seams of gas category I, presenting no dust hazards, the distance on strike between consecutively ventilated coal walls can be as much as 200 metres. In this instance, however, the number of successively ventilated walls should not exceed three;

"b) in gassy mines, the air entering each wall must not contain more than 0.5 per cent of methane;

"c) each successively ventilated coal wall is to be supplied with a supplementary current of fresh air from the adjacent haulageway (intermediary entry);

"d) when blasting is carried out in the bottom coal wall, the miners engaged in the consecutively ventilated walls lying above should be taken out to where there are main or supplementary fresh air currents;

"e) all consecutively ventilated coal walls with a total length of over 120 metres should be included in the general or internal telephone network."

Ventilation of two contiguous coal walls by separate air currents is shown in Fig. 217. The current flowing along the main entry near the slope is split into two currents at point *1*. One sweeps the bot-

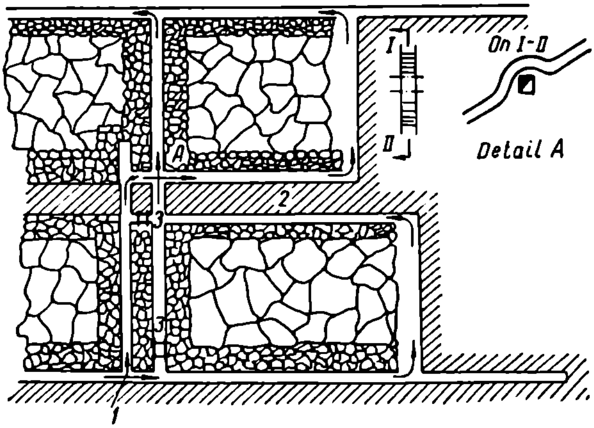



*Fig. 217.* Separate ventilation of two coal walls

tom coal wall; the other moves down the slope, then along the intermediary entry and finally circulates along the area near the face of the top coal wall. To split the air currents, coal pillars *2* are left between the walls. The currents are directed by seals or stoppings *3-3* with double ventilating doors. Where air currents cross each other (point *A*), air bridges have to be constructed. Coal pillar *2* may be replaced by a pack wall built with clay mortar.

In the mining of beds evolving firedamp, the continuous method has a number of major advantages and disadvantages. One advantage is the almost complete absence of dead faces and openings driven in advance of stoping, for it is in such dead headings that mine gas is likely to accumulate in dangerous quantities. But, as it often happens with room-and-pillar methods of mining, the development headings driven ahead of coal faces cut the solid mass of coal into separate parts (pillars), which emit most of the firedamp before stoping is begun. In other words, the network of development openings plays the role of drainage for the gas. Furthermore, in continuous mining the air current sweeping the workings of the entire level is either not split into separate secondary currents or is split much less than in the case of the pillar systems. In the final analysis, continuous mining may be regarded as quite suitable for working gassy seams.

## 4. Continuous Mining with Raise and Oblique Faces

Before the wide introduction of coal-cutting machines and combines, continuous faces sometimes were advanced *diagonally* (Fig. 218) or even very close to the direction of strike (Fig. 219). In the latter case, the faces advanced up raise and were therefore called *raise faces.*

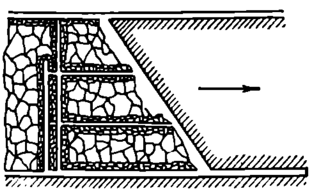


*Fig. 218.* Continuous mining with oblique coal faces

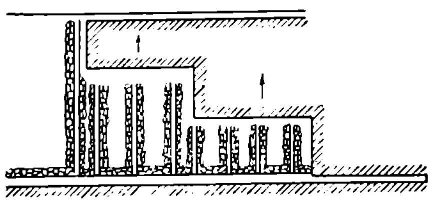


*Fig. 219.* Continuous mining with raise faces

In both instances such disposition of coal walls was dictated by a desire to facilitate breaking coal with hand tools or pneumatic hammers.

At present this practice has fallen in disuse because it is utterly inconvenient for the commonly practised methods of mechanised mining. In faces of this type coal combines, cutters and conveyers operate in an oblique position.

## 5. Continuous Systems in Mining of Slightly Inclined and Sloping Seams

The *longwall* variety of continuous mining in conditions distinguished by an easy and moderate dip is applied in working seams of low and medium thickness which occur regularly and without geological dislocations. Inasmuch as a continuous face is worked by combines and coal cutters, coal in the bed may be hard. The wall rocks should be steady or of medium rigidity. Thanks to its simple ventilation, the longwall system can be employed in working gassy seams, if the volume of the air supplied and the velocity of its current allow for the length of a coal wall determined by the level interval. This method is also suitable for mining beds with self-igniting coal, since no coal pillars are abandoned in the mined-out area in this case.

Generally speaking, this system should be chosen whenever the nature of the occurrence and wall rocks of a seam make longwall mining technically feasible. Fully provided with the necessary mining machines and with adequate organisation, longwalls yield the best results in coal production.

The situation is radically different in the case of continuous mining in coal deposits *divided into sublevels.*

As pointed out above, this system has a number of major disadvantages: 1) difficulties in making slopes in goafs; 2) high maintenance costs in intermediary entries and slopes in goafs; 3) absence of development openings running in advance of working places, precluding preliminary exploration of conditions of the occurrence of the seam. Hence the possibility of unexpectedly encountering geological disturbances which disrupt normal stoping operations.

These disadvantages are not inherent in mining with long pillars (Chapter XIII) and it is this method that should be used in working slightly inclined and sloping beds when conditions rule out mining by longwalls.

An exception in favour of continuous mining in its sublevel form may be made only for extremely thin seams (less than 0.7 metre) with regular occurrence, when openings can be satisfactorily maintained in goafs and protected by pack walls. Furthermore, continuous mining is preferable in working beds with bulging rock walls, as well as seams dangerous by their sudden outrushes of coal and gas.

Finally, note should be taken of the following fact. In recent years the Donets coal fields have made wide use of the damaging practice of servicing each coal wall, not infrequently only about 100 metres long and less, by a special haulageway, as a rule, with electric haulage, that is, without intermediary slopes. This has led to unnecessary scattering of work places in the mine and increased the length of the openings maintained. Now that there is a justified tendency to increase level intervals, these shortcomings should be eliminated and for that the question of applying continuous sublevel mining in individual cases should be raised again, provided the conditions stated above are complied with.

Continuous mining reduces coal losses to the minimum (to 3-5 per cent).

## B. STEEPLY PITCHING SEAMS

### 6. Development Work

When the pitch is heavy, continuous mining acquires specific features.

A general aspect of continuous mining with longwalls without sublevels is shown in Figs 190 and 226, and with division into two sublevels in Fig. 227.

When cap and floor coal pillars are provided over and under the entries, the openings are made in the shape of monkey entries with timber sets (Fig. 220), or else have composite timber and metal supports (Fig. 221), or only metal supports (Fig. 222).

The support shown in Fig. 221 is called *joint-shaped arch*. Under the impact of rock pressure it is capable of slightly changing its form. In a larger measure this property is inherent in the so-called arch compressible support (Fig. 222). The latter consists of three steel members with a special channel section. The butt end of one is inserted into the other and the juncture is tightly braced by two clamps, this causing high friction between the two members. It is for this reason that the support can withstand considerable loads before it becomes pliable.

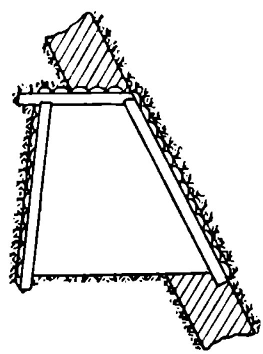


*Fig. 220.* Timber-supported entry in a steep bed

To avoid any disturbances of roof rock continuity, slashing is done at the bottom of the seam. In the case of ordinary timber sets (Fig. 220), one post is set up obliquely along the back of the seam, while the head piece rests not only on the posts but has one of its ends inserted into a hitch cut out in the rock of the foot wall.

Abandonment of cap pillars over the entry causes losses of coal and this, in mining seams containing spontaneously igniting coal, may lead to underground fires. In the latter case, *pillars* of barren rock obtained from driving the entry are erected over the opening instead of coal pillars (Figs 223 and 224). To protect the posts from the pressure of the waste pillar, an *overlap support* is set up at the height of about a metre above the entry. This consists of twin props, some of them in normal position with regard to the bedding and others serving as struts for the former (Fig. 224). The props have a stage with a slab flooring to hold broken barren rock. The waste pillars between which interstices are left to let the coal down from the production face are held in place on their sides by a sheathing

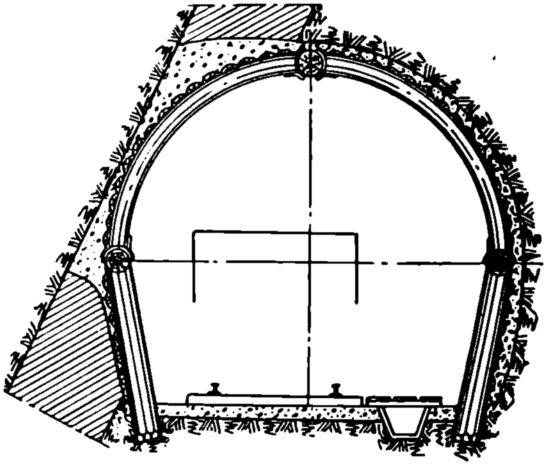


*Fig. 221.* Entry in a steep seam with a composite joint-shaped arch support

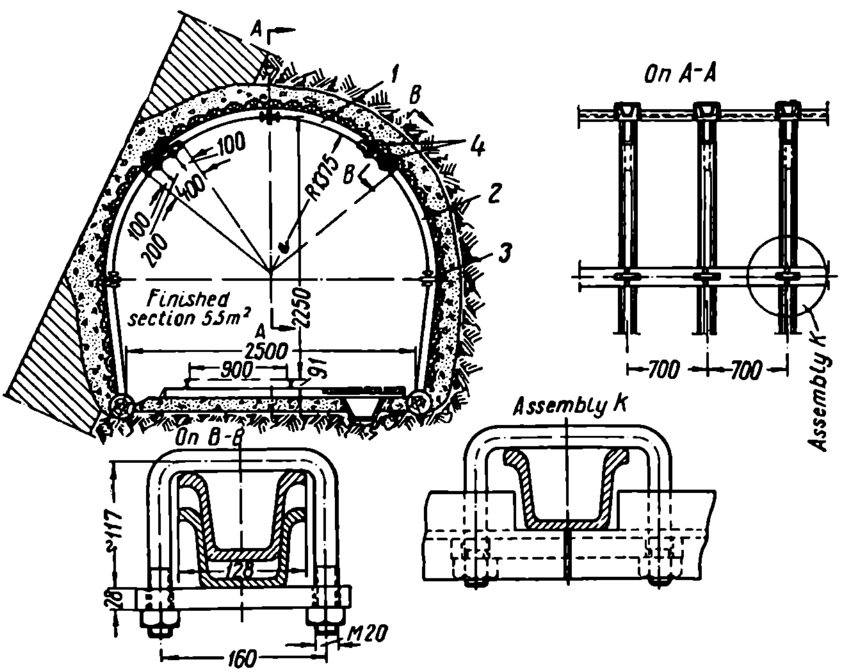



*Fig. 222.* Compressible arch support in an entry
*1*—top section; *2*—bottom section; *3*—longitudinal frame brace; *4*—key

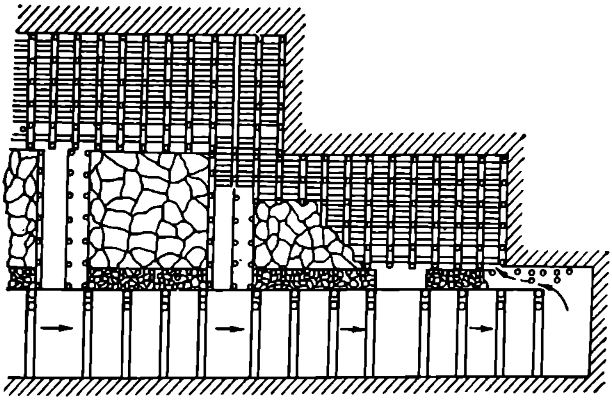


*Fig. 223.* Waste pillars over an entry in the mining of a steep seam

of slabs nailed to the props. The waste pillars are usually about 3 metres high.

When there is heavy side pressure, a lateral *strut* is put under the head piece or cap of the timber set (Fig. 224).

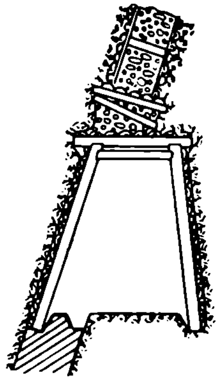


*Fig 224.* Entry support in a heavily pitching bed

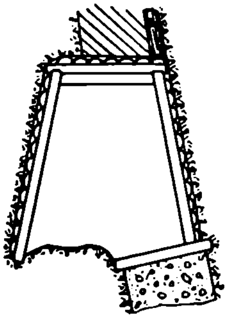


*Fig. 225.* Support in an entry with solid fill underneath

If no floor pillars are left under the airway, the props of the timbering along the hanging wall are set on struts too (Fig. 225), since the waste pack then would not be a sufficiently reliable foundation.

## 7. Longwalls in Steeply Pitching Beds

When there is a single continuous face in the level in a heavily pitching bed, such a system should be regarded as a variant of longwall mining. When coal is extracted by combines, coal cutters or coal ploughs, the face is advanced along a straight line. When it is pneumatic hammers that are used for breaking coal, the continuous face in a steeply pitching seam is worked by the *overhand* method.

Sloping in an overhand face has already been described in Chapter XI and here we shall give only some supplementary explanations.

In heavy-pitch conditions, the fill under the upper entry (Fig. 226) is quite often built of waste obtained in repairing and retimbering underground workings. As already stated, a stage is arranged in the open goaf to receive and hold this fill.

In the case of heavy pitch, coal broken off in benches rolls by gravity down the face and along short slopes (rise entries) between coal or waste pillars, and thus can be loaded into mine cars via chutes. To reduce the amount of fines at the face, it is better to direct the flow of coal through *ground chutes* (Fig. 226). They are made of boards or thick slabs nailed to props from top to bottom. In the lower portion of the level, ground chutes are not brought up directly to the coal or waste pillars. This is done to reserve room for coal accumulating during mining, since it is not discharged through drawing chutes for further haulage in the mine uniformly, but as empties arrive. In order better to direct the flow of coal towards the drawing chutes, the bottom sections of the ground chutes are sometimes set in upright position, and complementary ground chutes, forming an angle with its vertex up (the so-called *caps*), are installed over the pillars.

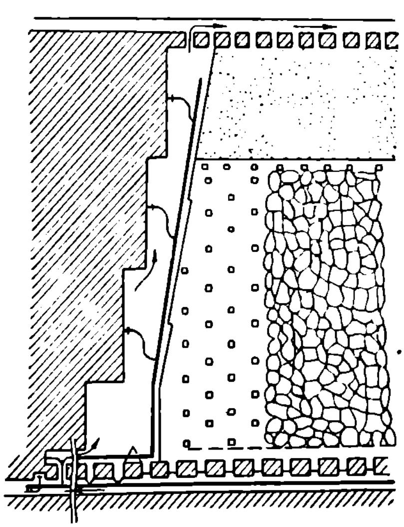


*Fig. 226.* Mining of a steep seam with partial filling

From the ground chutes coal goes to the dumping chutes between coal and waste pillars, whence, whenever necessary, it is dumped into mine cars. The minimum distance from the bench with ground chutes is 1-2 metres. As the working faces advance, the ground chutes are shifted, this usually being done every 4-6 metres. Since only thicker gangue intercalations are cut out separately in bench mining, while the thinner ones fall onto the ground chutes together with coal, and the latter, moreover, is contaminated with lumps of wall rocks, the ash content in coal won in steeply pitching seams is, all other conditions being equal, higher than in the case of gently sloping beds.

Sliding down the ground chutes, coal breaks up (the size of lumps depending on its hardness). To reduce the rolling speed in steeply set ground chutes, cross planks are nailed to them every three metres.

The special measures applied in the mining of coal seams which are subject to *spontaneous outbursts* of coal and gas (Chapter IX, Section 12) are governed by the Safety Rules. In reference to steeply pitching seams they are essentially as follows.

A coal measure which includes seams that are hazardous by their sudden outbursts should not be mined before a *protective* seam is drawn. The protective category includes beds occurring at a distance not exceeding 35 metres perpendicularly from dangerous seams. Stoping in protective beds should precede the advance of faces in the dangerous seam by at least double the distance between the seams in question.

Before approaching a bed liable to sudden outburst, a number of exploring boreholes not less than 5 metres long should be drilled from the heading of an opening (for example, a crosscut). Direct crossing of a seam by an opening is effected by blasting a solid mass of rock not less than 1 metre thick. The blaster should be at least 200 metres from the breast of the face, in the way of an intake air current.

Sudden rock falls during the driving of development workings can be forestalled by setting up advance timbering, drilling exploring boreholes and, in firm rocks, by shock blasting. The heading of a haulageway should be at least 50 metres ahead of the production face. The driving of uphill openings should be preceded by the drilling of large-diameter holes with a heavy boring machine. The breast of the face should be constantly supported and tightly lagged.

In steep beds the bench should be as high as possible and the advance kept down to the minimum. The bench overlap is to be lagged all the way by slabs and boards, and supplementary posts and inclined struts should be set up in addition to usual face timbering.

## 8. Continuous Sublevel Mining in Steep Seams

When roof-pressure control in single-stepped coal face in a level is difficult, the level can be divided in two or more sublevels. In the latter case, the general front line of production faces in the level is not straight and it is easier to prevent wall rocks from caving in and sliding. Coal broken down in the wall of the upper sublevel is transferred to the lower sublevel through the intermediary entry and rolls down the ground chutes together with coal drawn from the benches of the latter.

This method of mining, however, has a number of major disadvantages: 1) coal transportation is complicated and the mineral breaks; 2) communication with the intermediary entry is possible only through working places and this presents many inconveniences; 3) ventilation of the upper sublevel is rather difficult.

These shortcomings are mitigated by the employment of *advance slopes a* (Fig. 227). These slopes or dumping chutes, arranged every

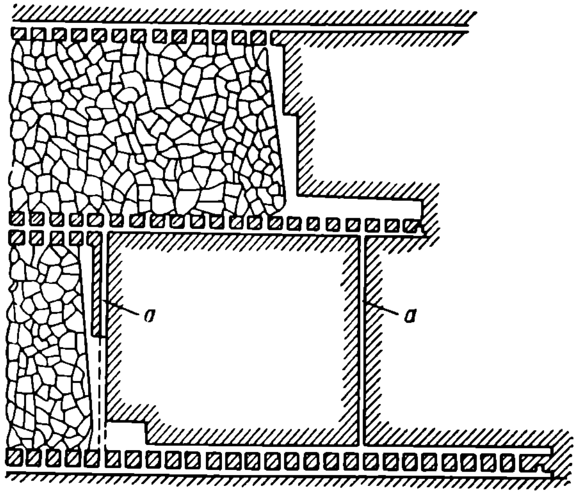



*Fig. 227.* Continuous sublevel mining of a steep seam

50-100 metres, are utilised for the dumping of coal, delivery of timber and equipment, passage of men and ventilation. Coal delivered to the intermediary entry from the upper sublevel is transported to the slope in mine cars or by conveyers. There is less breakage and dust production here than in ground chutes. The slopes can be used for supplying fresh air to the sublevels lying above. The slopes consist of dumping and manway compartments. The manway compartment is separated from the coal-dumping chute by a strong, solid partition with windows with bolts to allow jammed coal through. The windows should be 0.2×0.2 metre, with 3- to 5-metre intervals between them.

Mining with slopes is safer than without them, for in the event of a collapse of the roof or wall rocks in the production face the workers can find shelter in an intermediary entry, whence they can escape along the slope to the lower haulageway.

The use of slopes has its negative aspects too. It requires a greater amount of development work, and mining in coal faces, when they cross slopes, presents considerable technical difficulties. At these points coal is usually undermined on three sides and this may lead to its caving and to the downfall of "settled" rocks near the slope.

In view of the availability of slopes, the method above can be regarded as a *combined* one—a crosscut between continuous (long-wall) and pillar systems of mining.

## 9. Stand-By Coal Face Front

When a coal face, especially a long one, becomes temporarily inactive under any method of mining, it is liable adversely to affect the output of the entire mine.

Every coal wall produces considerable amounts of coal and the greater its share in the total mine output, the more its sudden shut-down affects the production programme of the mine. A coal wall may be put out of commission not only by the collapse of the roof, but also by some geological dislocation. Therefore, apart from the active faces, there should be stand-by faces and, particularly, coal walls.

To be really a stand-by unit, capable at any moment of replacing an active wall suddenly put out of operation, a wall must always be kept in good condition, that is, it should have solid timbering and the necessary machines and equipment. It is universally known that when a working face is stationary, the rocks in it become "settled", fissures appear, the boundary layers or slabs of rocks come off the back and fall to the floor, the posts begin to lose their initial "play", the cribs compress, etc., and not infrequently the bottom commences to bulge and swell. The nature of these phenomena becomes particularly adverse in wet workings. In a coal face it may be unsafe to resume stoping operations in such conditions. Therefore, a stand-by wall should periodically be "refreshed" by being advanced over a distance of several cuts.

The number of times a face requires "refreshing" depends on the firmness of rocks, quality of support and water-bearing capacity. Hence the different methods of maintenance of stand-by faces. When the rocks are hard, stable and dry, a well-supported stand-by coal face remains in good order for a long time and coal production therein may be resumed only in the event of trouble in active coal walls. A stand-by face surrounded by country rocks of medium stability should be "refreshed" systematically, at intervals determined by experience. And, finally, when the conditions are unfavourable for the maintenance of an inactive face, the notion of stand-by or reserve front of work places implies a greater number of active coal walls than is actually needed, to ensure planned output. In this instance, however, they should not all be operated simultaneously. An operative schedule of alternate mining of walls is elaborated in this case to enable the sum total of the active face front to ensure planned coal output at any given time. This mode of mining provides for the continued "refreshing" of all coal faces in the walls.

According to the Exploitation Rules the stand-by coal faces should account for at least 25 per cent of the total workings.

The disposition of stand-by walls depends on the adopted method of mining, the number of working seams and the sequence of extraction. When the beds are not all worked simultaneously, the stand-by walls can be located in seams which, though developed, as yet have no active working places. In this case, particular care should be made to avoid undermining these seams (see Chapter XXII). Stand-by coal walls may also be kept in developed but not yet mined sections of a seam. In the instance of coal measures, stand-by walls must not be provided in all of the seams, but only in those which are suitable for this. And, lastly, when coal walls "alternate", as described above, stand-by faces are concentrated within the limits of a single mine section.

In conclusion, we should emphasise that the availability of stand-by coal faces must by no means serve as an excuse for slackening attention in working active walls. Every effort should be made to ensure normal and uninterrupted operations in active coal walls, and that irrespective of the stand-by walls.

CHAPTER XIII

# PILLAR METHODS OF MINING

## 1. Main Points of Pillar Mining

The feature that chiefly distinguishes pillar methods from continuous mining is that in the case of the former, before stoping is proceeded with, the solid mass of the mineral in place is divided by development openings into separate parts, the so-called *pillars*, which are later recovered or robbed during stoping operations. The pillars are rectangular. In the instance of pillar methods, *the development workings are surrounded by coal pillars only and not by mined-out areas*, as is the case in continuous mining. Therefore, *there is no widespread subsidence of the roof over development openings in pillar mining*, they are subjected to much lesser rock pressure and, consequently, timbering requires less repair.

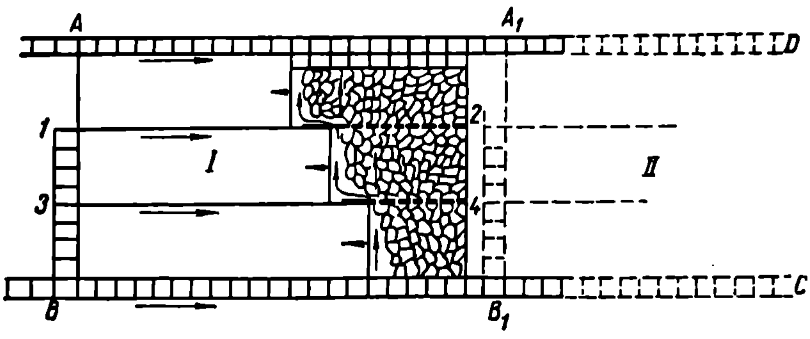



*Fig. 228.* Long pillar mining on strike

The pillars may be shaped like rectangles, usually extended *on strike* (Fig. 228), or, less frequently, *up raise* (see Fig. 241). There may also be pillars shaped like squares, called *short* pillars (see Fig. 242).

Hence pillar mining includes:

1) long pillar method on strike;
2) long pillar method up raise;

3) pillar-and-bord method.

The pillar methods can be applied in mining seams dipping at diverse angles.

### A. GENTLY PITCHING AND MEDIUM-STEEP SEAMS

## 2. Long Pillar Mining on Strike

As in continuous mining, development work is started from the air connection, inclined shaft, permanent mine slope or incline.

The level is divided into *panels* or *working sections*, each serviced by its own slopes or dumping chutes and successively developed and mined outwards and inwards. The slopes may be *unilateral* (Figs 228 and 229) or *bilateral* (Fig. 230).

From one of the above-mentioned permanent openings (*AB* in Fig. 228) several level drifts are run simultaneously—haulageway *BC* and airway *AD* and also intermediary (sublevel) entries *1-2* and *3-4*. The sublevel interval is determined by the firmness of wall rocks, the thickness of the bed, the angle of pitch and the adopted method of mining, and usually ranges between 80 and 150 and even more metres.

Intermediary entries extend only along the first panel $AA_1$, $BB_1$. The driving of intermediary entries cuts up the panel into long pillars on strike $AA_1$-*2-1*; *1-2-4-3*; *3-4*-$BB_1$.

Nothing will change if working *AB* is not one of the above-mentioned permanent openings, but some intermediary slope.

One can proceed with the extraction of coal as soon as the pillars in the panel are *cut* (some of them, at least). Recovery of pillars should be started in the *upper* sublevels (Fig. 228). Coal drawn from working places can be delivered along sublevel entries either to the *back* (Fig. 228) or the *front* (Fig. 229) slope.

Let us consider both cases in more detail. Recovery of pillars in a panel should *advance* at definite intervals, for a broken line of faces better supports the back.

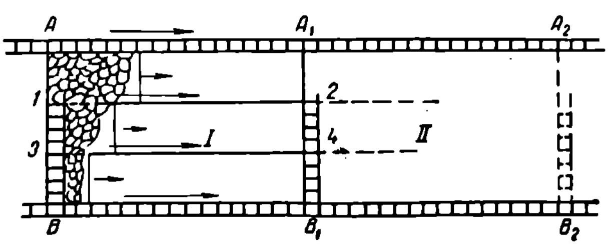



*Fig. 229.* Long pillar mining on strike with coal hauled to the front mine slope

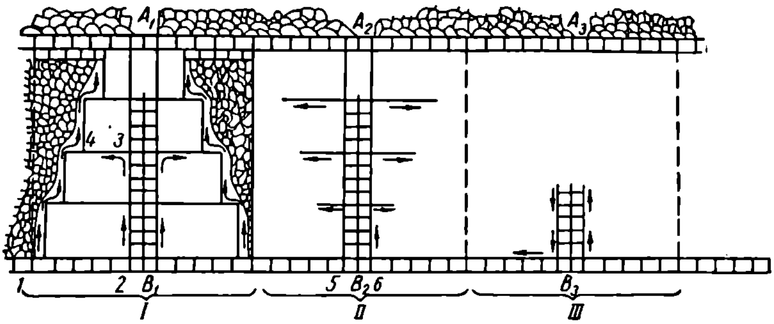



*Fig. 230.* Long pillar mining on strike with bilateral mine slopes

The top pillars are mined first for the following reasons:

1. Mining of the upper sublevels is started and, consequently, completed first, and the intermediary slopes are thus surrounded by intact pillars all through their service-life. This facilitates their maintenance. When the top pillar $AA_1$-*2-1* is mined, the length of the slope should equal *1-B*.

When the robbing of this pillar nears completion, section *1-3* of the slope may be abandoned, for it is only its portion *3-B* that is needed for dumping coal drawn from the pillar.

2. When water appears in working places, and the faces of the upper sublevels are run in advance, it flows to appropriate intermediary entries and through them to the manway or slope, both of which have drain ditches leading to the level haulageway.

The advance rate difference for these subentries is usually 10-20 metres.

As mentioned above, coal from production faces can be delivered either to the *back* or the *front* slope. In the first instance (Fig. 228), development openings in each panel are driven outwardly and stoping is done inwardly. There are ordinarily two active slopes on each side of the level: *AB*, which serves as a haulageway for coal extracted from the working faces of panel *I*, and $A_1B_1$, which services development work in neighbouring panel *II*.

When coal is brought to the front slope, each side of the level may be serviced by only one active slope (Fig. 229). For this, coal derived from work places is transported to slope $A_1B_1$. The latter is driven so as to be ready to operate by the time entry *1-2* has been driven and, consequently, the development of top pillar $AA_1$-*2-1* is nearing completion. It is at this moment that one can start recovering this pillar and then mine all the other lying below as the faces progress at the individually accepted rate of advance. The coal brought from the work places will be delivered to slope $A_1B_1$ on one side (Fig. 229, on the left). This slope, at the same time, will also be used for making development openings in neighbouring panel *II*. In this case, coal (and in some instances waste also) from the faces of intermediary entries will be hauled to slope $A_1B_1$ from the other side (Fig. 229, on the right).

Thus, intermediary entries are driven from slope *AB* to panel *I*, while the pillars prepared by them are mined through slope $A_1B_1$, which also serves the purpose of developing pillars in panel *II*, etc.

This mode of mining, however, presents considerable inconveniences:

1. Coal hauled through intermediary entries to the front slope and then brought along the slope to the level entry travels first from the shaft, and then again covers the same distance in the opposite direction, that is, to the shaft. This unnecessarily lengthens the haulage distance.

2. Development work and stoping in a panel proceed in the same direction, following the pattern of advance mining. But coal from the headings of development openings and from the coal faces is transported in different directions: in driving entries—to the shaft, in stoping—from the shaft.

Long pillar mining on strike with *bilateral* slopes is illustrated in Fig. 230. Here the slopes servicing panels *I*, *II* and *III* are not run along the panel boundaries, but along their centre lines. Intermediary entries are driven within the bounds of their panels, on both sides of each slope, that is, halfway between the neighbouring slopes. These are made to develop pillars, whose robbing proceeds in the direction from the panel boundary to their respective slope, as shown in Fig. 228.

Hence, in the case of the method described, intermediary entries are pushed forward from the central slope towards the boundary of the panel, while actual stoping is done in the opposite direction.

Level entries are driven either with cutting pockets in coal or on a narrow front, the latter method often being preferred because it makes the openings more stable. To provide protective sill pillars over the level entries a parallel opening is cut either without slabbing (a *through-cut*) or with it (entry), connected with the main entry through *crosscuts*. The making of through-cuts and crosscuts is usually called *blocking out* or subsidiary development of pillars. The size of protective pillars ranges very widely—from 10 to 40 metres on strike and from 10 to 20 metres on dip, depending on the properties of wall rocks, the cross-section of the entry and its service-life. These pillars are meant for protecting entries from the collapse of rocks capping the goaf for a period double that of the level's service-life, for each level entry serves both as a haulageway for a given horizon and as an air course for the underlying level.

Sill pillars over the haulage entry seriously inconvenience transportation of coal by conveyers from the lower sublevel, since it is impossible to load coal directly from conveyers into mine cars spotted along the haulageway, and a supplementary conveyer has to be set up in the through-cut or in the parallel entry.

To avoid these drawbacks, sill pillars are sometimes not left even in pillar mining, and the haulageways are protected by a rib fill of waste obtained from a dummy roadway pushed forward in the wake of the coal wall.

Intermediary slopes are driven either on a narrow front or with pockets cut out in coal. In the latter case, when rocks are relatively weak, the slopes with their manways and the abutting intermediary entries are sometimes driven on a narrow front at their intersections. The slopes are driven to the airway or the intermediary entry. The slope manways, however, should be run up the entire length of the level interval to ensure communication between the haulage entry and the ventilating course. In unilateral slopes the manways leading to the coal faces are so built that the men going to and from work do not have to cross the slope. In bilateral slopes the manways usually flank them on both sides (Fig. 230). To avoid transporting barren rock during the cutting of intermediary entries, they are almost invariably driven with pockets cut out in coal, not infrequently with a gateway or slant left. To ensure better ventilation, they are quite often arranged jointly with a through-cut (break-through) and crosscuts.

Intermediary slopes are almost always run from the level haulageway to the ventilating entry. If the level interval is considerable, this method may create difficulties in that development openings in long slopes have to be driven far in advance of coal faces.

Thus, for example, with stoping going on in panel *I*, development workings are excavated not only in panel *II* but in panel *III* as well.

In seams producing large amounts of firedamp there is yet another inconvenience—gas is very difficult to remove from the rise headings of the slope and its manways. Therefore, *when the level interval is considerable and there is appreciable evolution of firedamp, intermediary slopes $A_1B_1$, $A_2B_2$, etc., are sometimes started as inclines not from the lower but from the upper level entry down the dip.* It is quite natural that in these conditions coal must be brought up to the airway and special rooms are cut out for that at points $A_1$ and $A_2$ to accommodate air or electric hoists. Coal can be hoisted up by belt or flight conveyers. When water appears in the workings, a special pumping plant has to be provided for. Ventilation here is a much simpler matter because, being much lighter than air, firedamp leaves the stopes and ascends to the airway. Another merit of this method is that, proceeding from an air entry, the panel can be developed irrespective of whether the lower entry has already been driven over the entire distance of $B_1B_2$ (Fig. 230), the only thing necessary is to time its completion to that of incline $A_2B_2$. Opening $A_2B_2$ serves as an incline only in the process of its arrangement, while during stoping operations it is used as a slope, that is, for the delivery of coal to the lower haulageway.

Pillars in continuous faces are *stoped* by methods described in Chapter XI.

The most efficient method of coal transportation through intermediary entries and slopes involves the use of *conveyers*. Since coal walls are also serviced by conveyers, this will help to achieve complete equipment of the mine's working sections with *conveyers*. Conveyers may be operated by *remote control*.

In pillar mining, *ventilation* of coal faces and development openings is much more complicated than in continuous faces (longwalls).

Let us now study the pattern of ventilation applied in mining with bilateral slopes (Fig. 230). The ventilating current enters an opening in the given level wing through the lower haulageway. It branches out at many points. At point *1*, for example, part of it goes along the crosscut to the work places on the left side of the panel, covered by slope $A_1B_1$. Usually this air current is not deemed sufficient to sweep all the faces, and it is supplemented at one or several points by currents supplied via intermediary entries. At point *2* part of fresh air is diverted from the main current, flows along the manway over distance *2-3* and then along intermediary entry *3-4*, where it again merges with the main air current. The two merged currents sweep production faces of the upper sublevels and then escape into the ventilating entry. The other side of working section or panel *I*, where stoping operations are in progress, is ventilated in similar fashion. A *permanent stopping* usually shuts out the passage of air from the bottom portion of the slope.

In discussing the adoption of the scheme of unilateral ventilation (successive aeration of working places), due account should be taken of safety regulations, cited above in Section 3, Chapter XII.

The remaining part of the main air current is diverted to the main haulageway to ventilate development openings. Secondary currents are split from the main one at points *5* and *6* and directed into manways at slope $A_2B_2$ to ventilate advance headings in the intermediary entries. Air passes into the headings proper via special parallel longitudinal through-cuts, or is directed by air partitions or through ventilating tubes at the expense of the total mine depression or, finally, through individual ventilation facilities in accordance with the *separate aeration* principle.

The last portion of air enters heading 7 of the main entry, returns along the through-cut, sweeps the heading of slope $B_3$ and its manways, again descends to the through-cut, flows along it and through one of the manways running parallel to slope $A_2B_2$, and escapes into the ventilating entry.

If there is a considerable number of intermediary entries, the air currents entering the headings must cover extremely long distances, and this makes adequate organisation of mine ventilation difficult. To reduce these distances, the pillars may be traversed by coal headings which, when necessary, can be arranged every few scores of metres.

When the air current circulating along the working places passes from one sublevel to another, to divert it in the desired direction when there are intermediary entries with pockets in the coal, passageways in the fill of these pockets are made every few metres.

In the case of long pillar mining on strike, separate ventilation of coal walls may be effected by measures analogous to those cited above (see Fig. 217).

From what we have said above it follows that *in working with long pillars on strike the problem of ventilation is rather complex.* It requires a large number of ventilating facilities (air doors, partitions, stoppings, ventilating tubes, etc.). Especially difficult is ventilation of stub headings or dead faces in gassy mines.

On the other hand, it is desirable to split the main current (unidirectional ventilation) because polluted or methane-saturated air then does not enter other workings. Besides, the depression required for the movement of the air current is appreciably smaller than in the case of a common, unsplit current.

The network of development openings pushed ahead of stoping operations is a factor which contributes to the preliminary evolution of large volumes of methane from the seam and its wall rocks and thus reduces its amount in the production faces.

### 3. Spatial Correlation of Coal and Development Faces

When stoping by the long pillar mining method is in progress in one working section, the neighbouring one or two sections or panels are in the stage of development. A definite *correlation* must obviously exist between stoping operations and development work, viz.: stoping should be started in a given section when development is nearing completion. Development should naturally not lag behind, for this might well delay the normal course of coal winning. On the other hand, it would be useless and even harmful to push development work far ahead of the planned schedule, that is, with a long time interval between termination of development work in a given section and commencement of actual coal extraction. This would entail considerable premature outlays and uunecessarily increase the cost of maintaining development openings. In estimating the correlation of development work and stoping operations, one should allow for a certain *time margin*, that is, development work should be finished several months before coal winning actually begins—to counter any unforeseen delays which might be caused, for instance, by geological dislocations.

The estimate itself is to be made with two principal aims in mind:

1) to determine the eventual arrangement of development workings at a given position of coal faces and the adopted time margin;

2) to establish the factual time margin available for the development work, proceeding from the actual position of coal faces and development headings.

Let us turn to the first task. To solve it one should have the following information on hand: a) mining plan of an active or projected mine or its section; b) adopted or envisaged order and sequence of driving development openings; c) advance rates of coal faces and development headings; d) time margin for making development and permanent openings.

The Exploitation Rules do not set any definite schedule for the arrangement of blocking-out or development workings, but with regard to permanent mine openings they stipulate: "The opening and development of any subsequent level or horizon (driving of permanent mine openings) should be effected well in advance so that all preparatory work needed to ensure total planned output is terminated at least five months before total stope footage in the active level is reduced in the instance of shaft deepening, and not later than three months in all other cases."

A greater time margin is recommended in the following cases: for more complex systems of mining requiring larger volumes of development work; for main openings, as contrasted to secondary ones and blocking-out workings and in mining of faulted deposits. In the latter instance, the advance rate of development headings over production faces is relatively greater than in a regularly occurring deposit so as to allow for the exploration of any possible faults, folds, pinches, squeezes, swells, etc., and to prepare new working sections or panels in conformity with the results obtained in the course of exploration.

The principle underlying the general approach to estimating a normal correlation between working faces and development headings implies the following: the mine plan of a section in the stage of mining envisages a time interval in the course of which stoping operations in any arbitrarily chosen coal face and in a working section or panel are to be completed. The simplest way is to take the coal face (though this is not obligatory) that will be worked out first.

When no time margin is envisaged, a new working face in the next panel has to be made ready in place of the one closed down before the interval mentioned above lapses, if the plan of mine output is to be adhered to. But since in any estimates of this kind a certain time margin is quite indispensable, the new production place is, of course, to be prepared for stoping beforehand, in accordance with the adopted time margin.

A definite order for the fulfilment of all development and blocking-out work needed to open a new production face is outlined in the mine plan. To determine the normal position of headings in these openings, a position which would correspond to the initial position of the coal face in the actually mined panel, we must estimate what blocking-out and development openings (and of what length) can be made in the section under preparation to allow establishing a new production face within the time interval remaining until the final exhaustion of the coal face under consideration in the actually mined panel. In calculating the time necessary for driving the above-mentioned openings, we proceed from the assumption that they are made in the order and sequence *reverse to the factual.*

*Example.* The problem is to establish a normal correlation between production faces and development headings in the system of mining illustrated in Fig. 231.

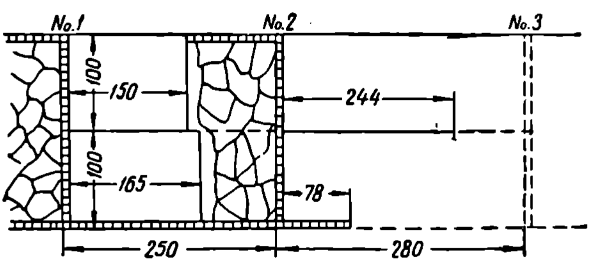



*Fig. 231.* Estimating the advance rate of development headings over that of production faces

The basic parameters of the method are given in the drawing. The extent of working sections or panels on strike, as it often happens in practice, is not exactly the same for all.

Let us assume that the slopes (with manways) are raised from the bottom, the intermediary entry is driven from the front slope and the through-cuts from a corresponding sublevel. The face advance rates are as follows (in metres per month): coal wall—40, level entry—88, sublevel entry—120, slope—60, longitudinal coal heading—88. The time margin for secondary openings (intermediary entry and coal headings) is two months and three months for the haulageway and slope.

Let us find the normal position of the development headings at the time when production faces are brought to the points indicated in Fig. 231, that is, when the coal wall of the top sublevel is 150 metres from the safety pillar near slope No. 1.

If we start from this moment, the top pillar will be recovered in $\frac{150}{40}=3.7$ months. Since the adopted time margin is two months, a new wall in the upper pillar of the next panel must be prepared for mining in 1.7 months.

Driving a coal heading (through-cut) 100 metres long will take $\frac{100}{88}=1.15$ months. Considering the time necessary to prepare the new coal wall for stoping, we increase it to 1.4 months.

Thus, the time left for driving the intermediary entry will be $1.7-1.4=0.3$ month. With the monthly advance of 120 metres, this heading can be pushed forward $120 \times 0.3=36$ metres during this time interval. In other words, at the moment under consideration the heading of the subentry should be $280-36=244$ metres away from front slope No. 2.

Let us now determine the normal position at this same moment of the heading of the level haulageway. Inasmuch as the intermediary entry is driven from the slope, the entry level must by this time be pushed forward to a point not nearer than slope No. 2. But since extraction of coal in the upper sublevel, covered by slope No. 1, will take 3.7 months to complete (barring the time margin for the time being), and complete recovery of the top pillar of No. 2 panel another $\frac{280}{40}=7$ months, the top coal wall of panel No. 3 must be prepared for stoping in $3.7+7=10.7$ months.

Development and blocking-out operations in panel No. 3 will last (calculated. as we have already said, in the order reverse to their factual execution):

| | | |
|---|---|---|
| Through-cut coal heading and preparation of coal wall | 1.4 | months |
| intermediary entry—240 : 120 . . . . . . . . . . . . | 2.0 | " |
| slope—120 : 60 . . . . . . . . . . . . . . . . . . . . | 2.0 | " |
| Total | 5.4 | months |

The available interval calculated without the time margin has been found to equal 10.7 months. By introducing the time margin (which is absolutely indispensable) we cut the time interval down to 7.7 months (since in our example the time margin for major development openings is accepted as three months). Hence driving the level entry till slope No. 3 may take $7.7-5.4=2.3$ months. With the accepted monthly advance rate of 88 metres, the heading of this entry will move $88 \times 2.3=202$ metres in this period. In other words, at the moment under consideration the heading of the haulage entry should be $280-202=78$ metres beyond slope No. 2.

By employing an analogous method, we find that this position of the entry heading can also fully ensure the timely development of panel No. 4 which, according to plan, is to extend 250 metres on strike.

We thus come to the following solution of the problem: with the production faces in panel No. 1 in the position indicated in Fig. 231, slope No. 2 should be completed over its entire length, the intermediary entry in panel No. 2 should be driven 244 metres, and the haulageway heading moved 78 metres beyond slope No. 2.

The second problem—determination of the factual time margin for development work—is solved even more simply. For this it suffices to compare time $t_1$, after the lapse of which the first coal face to be discarded in a given panel will have been actually closed down, with time $t_2$—the time required to start a fresh similar working face in the next panel.

All the calculations are done according to the mine plan and on the basis of the actual position of development headings, with due consideration for the adopted sequence of their driving and advance rates. The difference between $t_1$ and $t_2$ will represent the time margin available for stoping and development work. If necessary, similar calculations can be easily made with respect to other headings.

If the difference between $t_1$ and $t_2$ turns out to be inferior to the fixed time margin and, what is more, proves to be a negative value, it will mean that development operations lag inadmissibly behind and it is necessary to take urgent measures to remedy the abnormal situation.

Summarising, we see that the forcing of development work may, generally speaking, be effected in the following ways:

1) by increasing the advance rates for headings;

2) by revising the order accepted for making development openings. For example, in the instance of the mining method illustrated in Fig. 231, the future slopes may be arranged ahead of time if sunk as inclines from the upper entry. Such measures do not always prove technically suitable and economical, but they may be necessary to speed up development work;

3) sometimes, to force the development stage of operations, it is deemed advisable to change the planned parameters of the mining method. For example (Fig. 231), to accelerate blocking out a new panel when the driving of the level entry is delayed, a new slope is arranged at a point somewhat nearer than the one set.

There is, however, absolutely no justification for recommending measures connected with arbitrary deviations from the parameters of a mining method considered most suitable for the given conditions.

The above-mentioned method of estimating the advance of development headings is commonly used, for it can be applied equally well to any mining system and way of opening up deposits and winning any kind of valuable minerals.

## 4. Estimating the Size of a Working Section or Panel

Let us now consider the problem of spacing intermediary slopes, that is, of determining the size of a working section or panel on strike. By using the method of determining the size of a mine field discussed in Section 14, Chapter II, we shall find the most economically advantageous distance.

In other words, our task is to find the size $x$ of a working section which would reduce to the minimum the cost per unit of output (1 ton), according to items depending on this size. With this aim in view, let us sum up the corresponding expenses for the whole of the working section and, to charge them against a ton of output, divide the total by the sum of the workable reserves contained in the section:

$$Z = hxpc$$

where, as before, $h$ is the level interval;
$p$—average output per sq m of the seam in tons;
$c$—coefficient of recovery.

Of the cost items to be considered, we take only those incident to the excavation and maintenance of openings and to haulage along intermediary entries, and that for the following reasons.

The cost of stoping in the walls should be disregarded, for it has nothing to do with the size of the section on strike. Likewise, of no import for the solution of the problem is the cost of driving level and sublevel entries, since they are made irrespective of the spacing of slopes.

By $K$ we will designate the full cost of arranging one slope with its manways and level grounds at the crossings of these workings and entries—in a word, all the expenses incurred in arranging each new slope.

Let $q_2$ be the cost of "net" haulage of coal in intermediary entries, charged against a ton metre.

Further, let $r$ and $r'$ be the summary costs of maintaining a linear metre of entry per year in rubles during stoping and development operations. The annual advance rate of production faces let us denote by $L$ and that of intermediary entries by $L'$(m).

With these symbols the costs for the whole of the working section according to the items enumerated above will be

$$K + \frac{rx^2}{2L} + \frac{r'x^2}{2L'} + Z\frac{x}{2}q_2.$$

These costs per ton of output will be

$$f(x) = \frac{K}{hxpc} + \frac{rx}{2Lhpc} + \frac{r'x}{2L'hpc} + \frac{xq_2}{2}.$$

Let us rewrite this equation as follows:

$$f(x) = c_1x + \frac{c_2}{x}, \qquad (1)$$

consequently,

$$c_1 = \left(\frac{r}{L} + \frac{r'}{L'}\right)\frac{1}{2hpc} + \frac{q_2}{2};$$

$$c_2 = \frac{K}{hpc}.$$

Let us now find the size of a working section on strike $x_0$ under which the expenditure incurred per ton of output according to the cost items enumerated above will be minimum. To do this, we have to find the value for $x$ which reduces function (1) to the minimum, that is, equate its first derivative to zero:

$$c_1 - \frac{c_2}{x^2} = 0.$$

By solving the equation above we have

$$x_0 = \sqrt{\frac{c_2}{c_1}}.$$

And in detail

$$x_0 = \sqrt{\frac{2K}{\frac{r}{L} + \frac{r'}{L'} + hpcq_2}}.$$

This last formula determines the optimal intervals between slopes.

As we see, it depends on the cost of making slope $K$, maintaining entries $r$ and $r'$, haulage $q_2$, level interval $h$, output per sq m of the seam $p$, advance rates of headings $L$ and $L'$ and, finally, coefficient of net coal recovery $c$. Hence, the size of a working section or panel should be determined in each concrete instance, for its value is liable to change along with geological and mining conditions.

This problem can also be solved *graphically* if function (1) is represented to a certain scale in a drawing (Fig. 232). This graphic solution is especially illustrative of one of the most important aspects of the problem. The drawing shows that although there does exist an optimal value of function (1) when the size of the section is $x_0$, the unit cost per ton is very close to that recorded at the minimum point because the length of the section on strike is close in

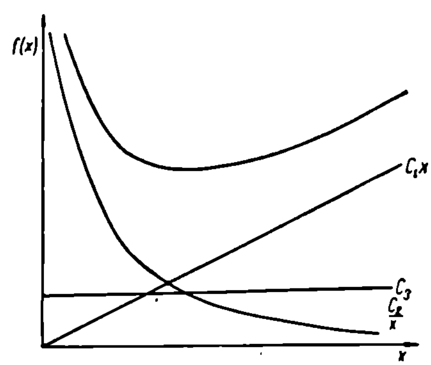



*Fig. 232.* Graphic method of determining the size of a panel

size to that of $x_0$. In other words, the size of the section determined by the formula above should be considered only as an approximation, as a guiding value in determining the optimal interval between two slopes. This inference is all the more true because all the values of the parameters included in the calculation, as well as the functional relations between them, are not exact but approximate.

The above-mentioned method of estimating the size of a working section or panel may also be applied in mining by other variants of the long pillar method, for instance, that involving bilateral working sections; in establishing spacings between slopes in steep beds, and with other methods of mining.

## 5. Long Pillar Mining in the Moscow Coal Fields

The conditions in which coal deposits in the Moscow basin occur are rather singular. Despite their Carboniferous age, the sedimentary rocks enclosing coal there are represented by little altered clays, sands and, occasionally, limestone. As far as their mining properties are concerned, in most cases they are weak, unstable and ready to cave in; the bottom is liable to bulge. Not infrequently the sands are saturated with water which, during inrushes into underground workings, may carry away large volumes of sand. In addition, in some places there are *quicksands*, both over and under the working seams.

The hardness of coal is variable, but in general it is rather weak and highly fissured. There is no firedamp in the now active mines, but frequent evolution of carbon dioxide tends to complicate ventilation of underground workings.

The opening of mine fields in the Moscow basin has already been described in Chapter II, Section 9.

The presence there of medium-thickness seams with rather weak coal occurring in unfavourable conditions had led to the adoption of pillar-and-bord mining (see Section 9), with pillars recovered by employing this or that modification of stub or butt entry method. But to make full use of the advantages of continuous face mining here too a change-over to long pillar mining with continuous coal walls has been effected in the past two decades.

There are several modifications of long pillar mining practised in the Moscow basin.

The system involving working of *single* walls (Fig. 233) implies mining of a panel or working section with one extremely long pillar blocked out by two butt entries and extracted in the reverse order. One of these two butt entries, driven on the side of solid coal in place, serves as a haulageway. Stoping in adjacent coal walls proceeds independently. Partial recovery of the coal pillar between the walls is effected parallel with the extraction of coal in the wall.

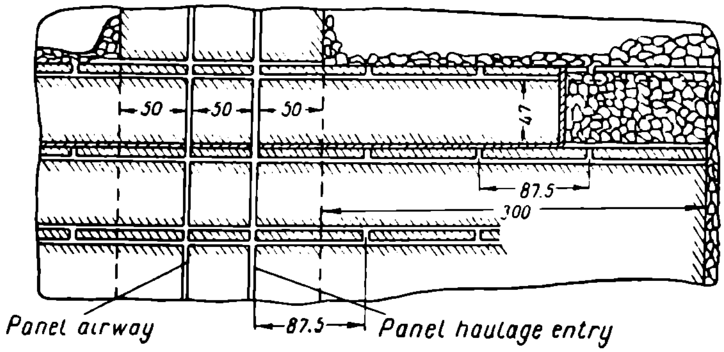



*Fig. 233.* Extraction of long pillars by single coal walls

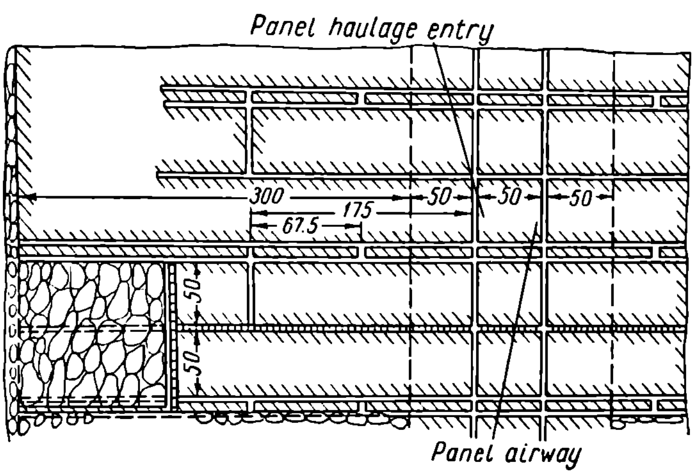



*Fig. 234.* Extraction of long pillars by twin or double walls

The system of long pillar mining with *twin walls* (Fig. 234) is a modification of working a panel with two long pillars blocked out by three butt entries and extracted in reverse order. The central, *mother* entry is used for the haulage of coal coming from both walls. The *outside* entries serve for the supply of mine timber. The walls are worked alternatively.

The same modification as above but with *independent* organisation of stoping operations, that is, simultaneous drawing of coal from both walls is known as long pillar mining with *double* walls.

Experience and technical and economic estimates give ample ground to assert that the best operative results in the Moscow basin can be achieved by applying the *double-wall* modification. In small-size, geologically dislocated panels of irregular shape with extremely complex hydrogeological conditions, it is more expedient to mine with single walls.

Because of weak rocks, the length adopted for each wall is of moderate order—around 50-70 metres, and the pillars extend over 300-350 metres.

Coal in walls is mined mainly by blasting, usually preceded by undercutting with the aid of a coal cutter. In certain instances, Donbas or BOM mine combines (see Fig. 147), or pneumatic hammers, are used.

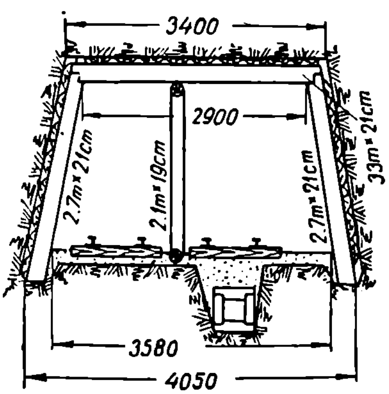



*Fig. 235.* Drain ditch

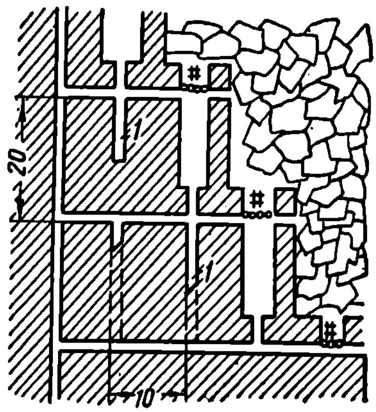



*Fig. 236.* Pillaring by stub entries

Stoping in the walls of the Moscow basin is distinguished by the pattern of timbering (described in Chapter XI, Section 3) employed due to the unfirmness of wall rocks.

Coal is transported in the walls by chain-and-flight conveyers, in haulageways by belt conveyers with flight conveyers installed near their butt ends to facilitate their shortening as the wall faces move on.

The Moscow basin is distinguished by its high water-bearing capacity and the level occurrence of beds with bottom horsebacks.

Therefore, the arrangement of ordinary outfall ditches is not everywhere feasible and special deep *drainages* have to be provided (Fig. 235). Dewatering of rocks can be effected with the aid of stemmed filters. In recent years preliminary draining of mine fields has been successfully done by deep-wall pumps.

Before the adoption of long pillar mining with continuous faces, pillars in the Moscow coal fields were worked by various modifications of the butt entry or shortwall method. One of them is illustrated in Fig. 236. Cross coal headings *1-1* were driven from panel entries spaced 8-10 metres from one another. From these cross headings coal was drawn bilaterally by shortwalls. Stumps of coal had to be abandoned near the goaf. The disadvantages of the shortwall method are the necessity of driving many openings with narrow faces and high coal losses. For this reason this method is now hardly used at all.

## 6. Long Pillar Mining of Combustible Shale Deposits

The principal sources of combustible shales in the U.S.S.R. are the Baltic (Estonia and Leningrad Region) and Volga basins.

Output of combustible shales and their utilisation in the national economy grow year by year. They are used as fuel, especially in powder form, for making gas, artificial liquid fuel and diverse chemicals. The large amount of ash which remains when shales are burnt may be employed for making bricks, slag-blocks and cement for construction jobs.

Despite their different geological ages, Silurian for the Baltic basin and Jurassic for the Volga basin, the conditions attending mining of shale beds in these two basins are practically identical. The beds are of complex structure and consist of a few benches of shale separated by intercalations of limestone (Baltic) and compact clay (Volga). The beds are about 2 metres thick. Both in the Baltic basin and in the important Kashpirsk district of the Volga basin

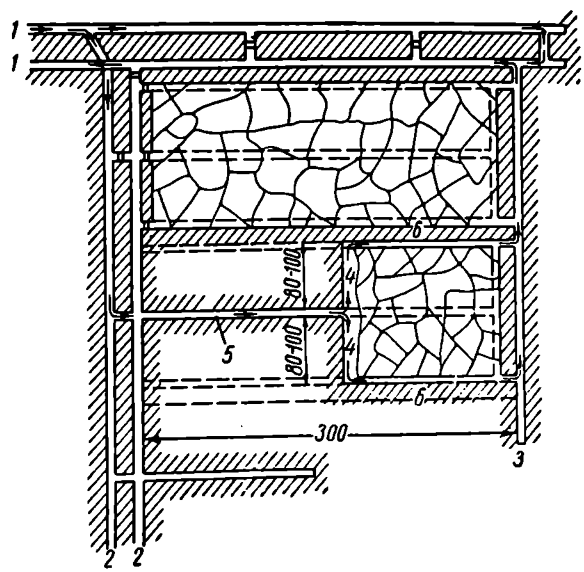



*Fig. 237.* Long pillar mining of a combustible shale bed

the beds occur almost horizontally. The roof and bottom country rocks are either firm or of medium stability.

These natural conditions favour long pillar mining with recovery of pillars by continuous mechanised walls (Fig. 237). Approximately every 300 metres panel entries—haulage *2*,*2* and ventilating *3*—are driven from the main entries. Their purpose is to block out the panels which are worked by twin walls *4*,*4*. The length of each wall is 80-100 metres. Shale drawn from production places is transported by conveyers to mother entry *5*. In the drawing the course of ventilating currents is indicated by arrows.

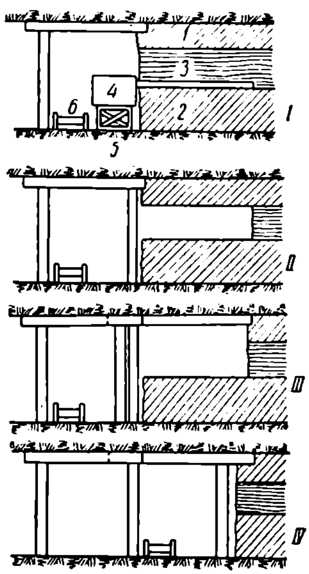



*Fig. 238.* Breaking a seam of complex structure

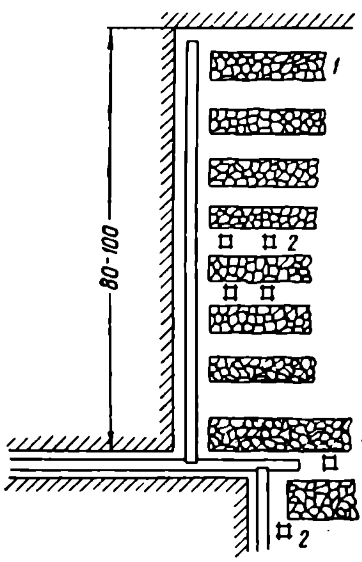



*Fig. 239.* Production face in mining combustible shale

As stated above, the beds of combustible shale are of complex structure. The relative position of shale bands and gangue partings preconditions the order of stoping. Fig. 238, for example, shows sequences *I*, *II*, *III*, *IV* of the undercutting and breaking of a bed comprised of two shale bands *1* and *2*, separated by gangue interlayer *3*. Since the cut is made in the interlayer to avoid excessive breakage of shale, coal cutter *4* has to be set on special slides or runners *5*. From the production face shale is transported by flight conveyer *6*. Breaking becomes complicated when the bed contains two or more gangue intercalations. Rock from the latter is stowed into the goaf. If there are large amounts of waste, the filling is complete, otherwise pack walls *1* have to be built (Fig. 239). When necessary, the back is supported by cribs *2*.

Fig. 237 is illustrative of a modification of the method under which twin walls are separated from the adjacent walls by boundary pillars *6*. If the outside, or boundary, entries protected by pack walls can be safely maintained in the mined-out area, no boundary pillars are left, thus reducing the losses of the valuable mineral.

### B. STEEPLY PITCHING SEAMS

## 7. Long Pillar Mining on Strike in Steep Beds

Section 8 of Chapter XII described a method representing a switch-over from continuous to pillar mining (see Fig. 227). However, when the pitch is heavy one can also apply the typical system of long pillar mining on strike with an overhand (Fig. 240) or straight face. This method is suitable for working steeply pitching seams of medium thickness when, for one reason or another, longwall mining is impracticable.

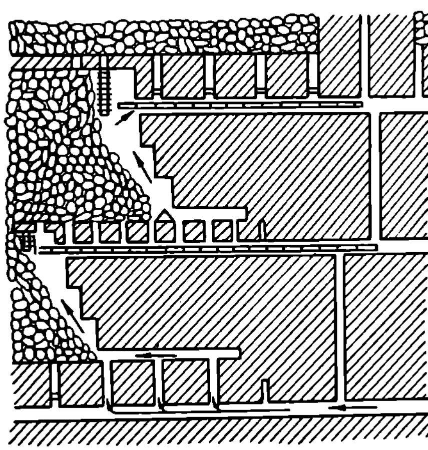


*Fig. 240*. Long pillar mining in a steep bed

From upper sublevels coal is dumped along the slopes. Intermediary entries are maintained in a solid mass of coal. This method (shown in Fig. 240), for example, is employed in the Kuznetsk coal fields. An analogous system adopted in the same region for working thicker seams is described below (Chapter XV). In the intermediary entries coal is transported by flight conveyers. To facilitate cutting dump chutes, special heavy boring machines are employed (Chapter XV).

## 8. Application Fields for Long Pillar Mining on Strike

This method has recently come to be very widely used, in many instances supplanting continuous mining.

In slightly inclined and sloping seams, long pillars stretching on strike are worked by highly efficient continuous, rectilinear faces. Stoping operations in these faces have been explained in detail in Chapter XI.

By its very essence, this method provides for driving development openings in panels prior to establishing any production faces. That, as we have already seen, offers a number of major advantages: 1) making of entries and slopes helps obtain additional information on the mode of occurrence of the bed within the boundaries of the panel; 2) development openings are maintained amid a solid mass of coal in place, which protects them from intensive rock pressure; 3) thanks to the network of development workings firedamp is drained prior to stoping operations in the panel; 4) when necessary, additional walls can be opened in prepared pillars; 5) the faces of development and production workings are independent of each other in pillar mining, and that simplifies organisation of operations in panels.

In the past, when there was no efficient equipment for driving development openings, the need for arrangement of intermediary entries, slopes and other workings was considered a shortcoming of the pillar methods. Today the problem is easily solved by adequate mechanisation of this process.

For example, there is the HK-1 combine, designed in 1952, for *primary coal mining* in slightly inclined beds 0.8-1.5 metres thick. It is equipped with automatic hydraulic controls and is capable of driving openings as high as the seam is thick and 3 metres wide. This machine makes it possible to drive openings to the rise, up the rise and on strike in beds with various angles of dip within the range of 25°. The daily rate of driving is as much as 8 linear metres.

The thinner the seam the more barren rocks have to be excavated together with the useful mineral in the headings of development openings. For this reason long pillar mining on strike may lose its advantages over the continuous methods in cases when the thickness of the bed is approximately less than 0.7 metre because development openings can then be maintained in the goaf (the same also applies to continuous mining) only if pack walls are built. However, if the beds are irregular, preference should naturally be given to the pillar method of mining, even if they are very thin.

The range of application of long pillar mining on strike is thus very wide and includes thin and medium-thick seams of up to 2-2.5 metres. A description of the method of long pillar mining on strike, employed in working thick beds, is given in Chapter XV.

This method can be employed in steeply pitching seams too.

Inasmuch as development openings in long pillar mining are maintained in solid coal, the system is applicable to wall rocks of different degrees of stability.

It has been already stressed that the necessity of providing for a wide-flung network of development openings calls for strict observance of precautionary measures against the accumulation and ignition of firedamp, especially in rise headings.

## 9. Long Pillar Mining to the Rise and Pillar-and-Bord Method

These methods, formerly widely used, are no longer of any import and we shall, therefore, discourse but briefly on them.

Fig. 241 shows that in long pillar mining to the rise development work is similar to that done in long pillar working on strike, except

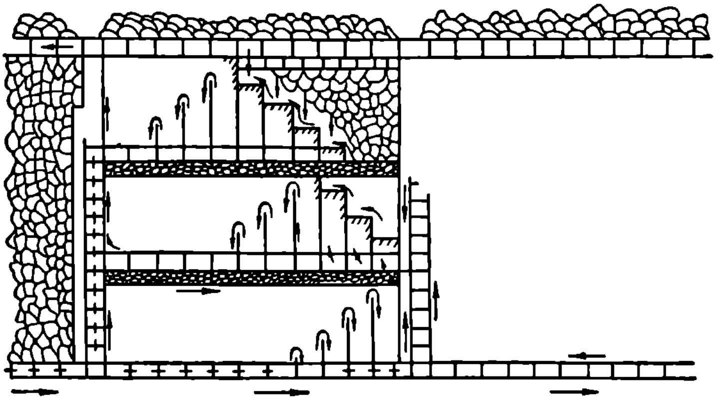


*Fig. 241.* Long pillar mining to the rise

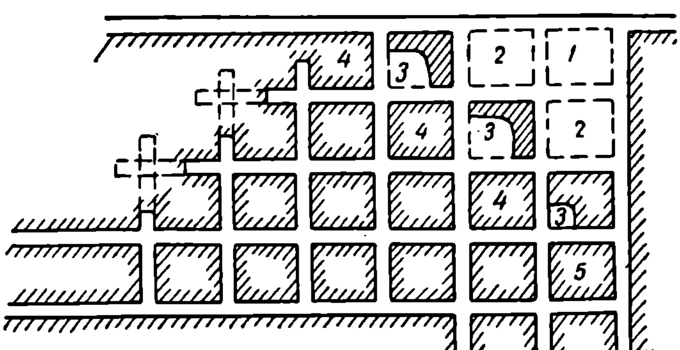



*Fig. 242.* Pillar-and-bord method of mining

that each sublevel is divided into *pillars to the rise* by *special raises*. Recovery of each pillar is started from the top by driving butt entries. From production faces coal is conveyed via the rise headings (raises).

This, and the fact that coal is drawn by shortwalls, explains why it is possible to use this system when wall rocks are poor. The pillars are blocked out gradually, as the need arises.

This method has many disadvantages: 1) it requires many openings to be driven with narrow faces; 2) production places are rather short, this making the use of mine machines difficult; 3) ventilation is an extremely complex affair; 4) the system is inadmissible in gassy mines in view of the danger of methane accumulating in rise headings; 5) scattered working places; 6) high coal losses.

The main principles underlying the pillar-and-bord method are illustrated by Fig. 242. As may be seen, this method is distinguished from the previous by the availability of *through-cuts* between coal headings and by the shape of pillars. The order of pillar recovery (usually by stub entries) is indicated in Fig. 242 by numbers. This system's shortcomings are similar to those of long pillar mining to the rise.

CHAPTER XIV

# COMBINED METHODS OF MINING

## 1. Basic Concept of Combined Methods of Mining

We already know two extensive groups of mining methods—*continuous* and *pillar*. The characteristic feature of the first is the absence of any development openings made in advance of production faces (except for level entries). In the second group, before the solid mass of the useful mineral is stoped, it is cut by development openings into pillars.

However, there may also be *combined* methods which possess features common to both the continuous and long pillar mining.

We are already familiar with the methods illustrated in Figs 216 and 227, which are a crosscut between continuous and pillar methods, and which, in this respect, may be included in the category of combined mining systems.

But, in the case of typical combined methods, production faces within the limits of blocks or panels are opened without primary or first mining in the solid mass of the mineral in place. With their progress the working block is cut into pillars, which are recovered at a later date.

Here we shall dwell upon two typical combined methods of mining.

## 2. Mining with Twin Entries

This method is illustrated in Fig. 243, where the position of mine workings in the first period is shown on the left, and the second period, marking the recovery of pillars, on the right.

Development and stoping operations with this method progress in the following manner. The level is divided into bilateral panels or blocks. Each side or wing of the block is worked out partly by the advancing production faces moving away from the slope

(Fig. 243*a*) and partly by the pillaring in the direction from the block boundary towards the slope (Fig. 243*b*). Working faces are opened via sublevels with intermediary entries following in their wake. Waste blasted in driving the entries is stowed into the open goaf over the lower and under the upper entries. Both entries are thus run simultaneously right after the common working face. Hence the name of the method—"twin entries".

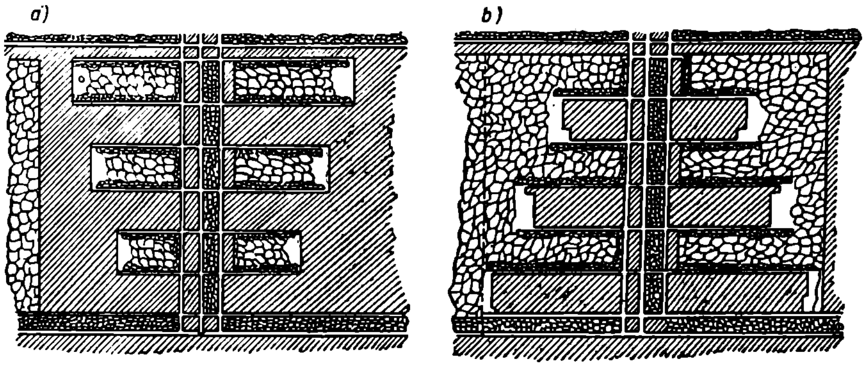



*Fig. 243.* Twin entry method of mining

To secure a sufficient amount of blasted rocks for reliable protection of entries from rock pressure, the thickness of the working seam should not exceed 0.8-1 metre.

When the headings of the even-numbered sublevels reach the boundaries of the block, the odd-numbered sublevels lying intact between them assume the shape of long pillars on strike. These are mined by retreating.

Among the advantages of the method under discussion are the possibility of an early start of stoping operations and the low coal losses.

On the other hand, the system has also its disadvantages:

1) production places are isolated from one another and have poor communications with the main entries;

2) entry headings closely follow the coal wall, thus creating considerable inconveniences for stoping;

3) with retreat mining of pillars edged on three sides by stoped-out areas, the support of the entries becomes subject to strong rock pressure;

4) the course of ventilating air currents is rather complex.

The author is of the opinion that, for these reasons, the method of twin entries is now devoid of any practical interest.

## 3. Room-and-Pillar Method of Mining

The basic principle of this method is illustrated in Fig. 244. *Rooms 1,1*, from 4 to 12 metres wide (usually 6-7) are run from the entries and separated by *rib pillars 2,2*, from 4 to 15 metres wide (more often 5-8). When the pitch is flat, the rooms can be excavated on both sides of the entries, 80-100 metres in each direction.

In order to form stub pillars near the entries, the rooms are first driven 6-10 metres with narrow faces (*neck of the room*) and are then enlarged to their full width.

To ensure adequate ventilation, the rooms are connected by breakthroughs *3,3* made in rib pillars. Each coal pillar between two rooms is mined by retreating towards the entry.

The combination of rooms and pillars helps solve the problem of roof control in an original way. The width of the rooms is determined on the basis of experience in a manner precluding the development of considerable pressures by back rocks. The nature of rock pressure bearing down on each room is similar to that experienced by any individual mine working. Overall pressure of the overlying rock masses during the mining of the rooms, on the other hand, is borne by the rib pillars. Hence, the nature of rock pressure and its control in room-and-pillar mining differ greatly from those in continuous faces.

Rib pillars are robbed by retreating and that causes high coal losses. It is only in isolated cases, when the back is very stable, that it is possible to work pillars by using "open end" method (continuous faces) *4*. More often, however, they have to be recovered by cutting pockets *5* (pocket-and-stump method), this entailing consider-

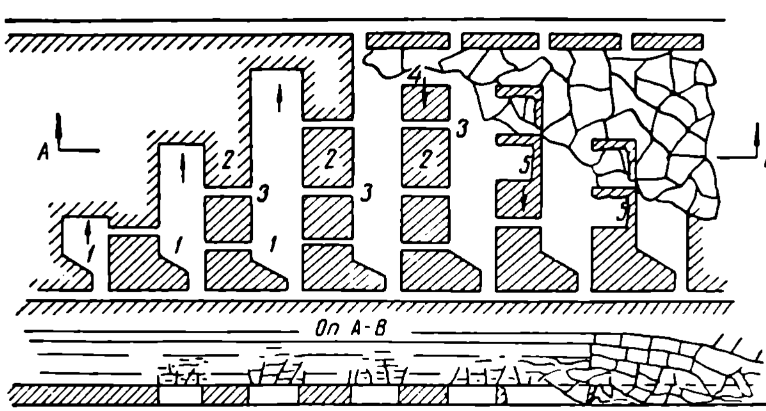



*Fig. 244.* Outline of room-and-pillar method of mining

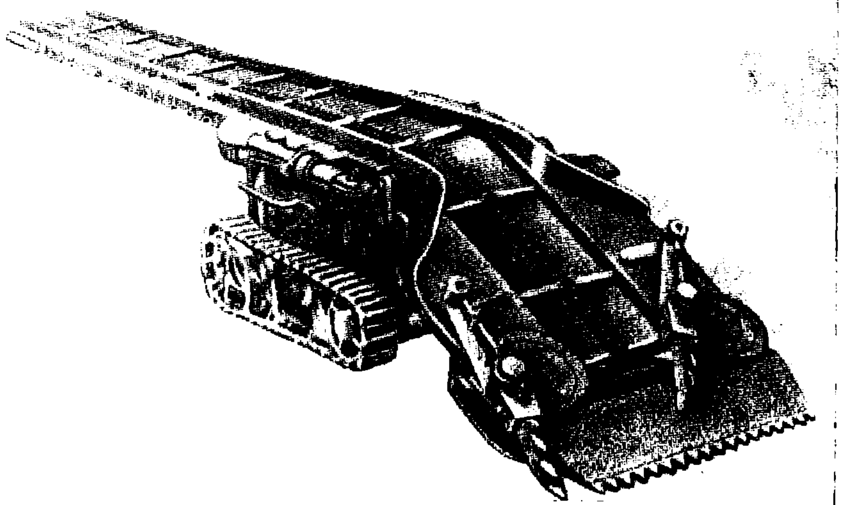


*Fig. 245.* Loading machine

able losses of coal in stumps. To ensure the stability of the roof it is essential to keep the faces straight. In some cases, depending on the nature of rock pressure in the rooms, weak timbering is used, in others there is no timbering at all.

As said above, rib pillars are worked back, by retreating. Since, during their recovery by pockets, coal stumps are edged on three sides by the goaf, their extraction is attended with large losses of coal, and that is the main drawback of the room-and-pillar method. Coal in rooms and rib pillars is undercut by coal cutters designed to work in narrow faces. After it has made an undercut in one of the rooms, this machine, which is set up on a special self-propelling carriage, is moved to other rooms, if the seam is flat or slightly inclined. The power driving the motor of the machine is supplied through a flexible cable.

Undercut and blasted coal is loaded directly into mine cars spotted in the room or onto a conveyer. The operation in both instances is effected by overloaders or loading machines (Fig. 245).

The room-and-pillar method of mining is very widely used in the U.S.A., where flat or very nearly flat rock occurrences predominate in most of the coal fields, the usual thickness of coal beds averaging about 1-2.5 metres and the wall rocks being firm and the seams regular. In such favourable mining conditions and with adequate equipment, the use of this method ensures very high efficiency of labour and low consumption of mine timber. Nevertheless, the room-and-pillar method of mining coal deposits in the U.S.A. is distinguished by extremely high losses of coal, scattered nature of working places and unsafe working conditions in gassy mines.

When beds occur at an angle of pitch, making transportation of coal complicated and, particularly, when frequent transfer of machines from one working face to another presents considerable difficulties (when each produces little), the efficiency of this system of mining drops sharply.

In the U.S.S.R., the room-and-pillar mining of coal seams has now come into an almost complete disuse and, it is apparently in exceptional cases that one can expect good operative results from its application.

CHAPTER XV

# MINING OF THICK SEAMS

## 1. Preliminary Observations

We have already seen that seams more than 3.5 metres thick are called high.

Generally speaking, the methods employed in working high seams are much more complex compared to those applied in mining low and medium-thick seams.

The methods of mining high seams can be classified into two distinct groups: a) those *without slicing*, and b) *slicing systems*.

The nonslicing methods are extremely variable, but by nature of development work they can be classified according to features analogous to those adopted in working low and medium-thick seams. True, in mining high seams the sequence of driving development openings is by far not sufficient to characterise the method, for the great thickness of the bed itself leaves its imprint on the modes of stoping that present the greatest variety.

Since full-seam or full-breast mining of thick beds is fraught with numerous difficulties, the slicing methods are very widely used. The underlying basic principle is that a high seam is not worked out at once over the whole of its thickness but gradually, by slices, so that the extraction of each such slice can be likened to mining a medium-thick seam. In space the slices may be horizontal or inclined (see below).

In view of the difficulties attending stoping, the *level interval* in the exploitation of high seams is generally smaller than in the mining of low and medium-thickness seams.

### A. NONSLICING METHODS OF MINING THICK SEAMS

### MINING BY "STRIPS"

## 2. General Concepts

We have already seen that the feature distinguishing the continuous mining widely used in working low seams is the small number of development workings driven ahead of production faces. In this sense the so-called working by *strips*, which is applied in the extraction of medium-thick and thick beds, is similar to continuous mining. This similarity, however, applies more to a coal block or panel, for certain development openings are first driven to separate these blocks within a level.

In essence, the method consists in extracting one *strip* or *band of coal* in the block by working faces, before starting to mine another. The working by strips can proceed on strike, up the raise, or diagonally.

## 3. Mining by Strips on Strike

A typical example of mining by *strips on strike* is shown in Fig. 246. Coal blocks are bounded by level or sublevel entries and inclined openings, which may be slopes or dumping chutes, depending upon the angle of dip. Mining by strips implies *complete filling* of stoped-out space.

Coal broken in the working places slides to the lower entry through the central dumping chute, while mine-fill from the upper entry is passed down through the side chutes on the boundaries of the block. Coal strips are extracted upwards. The width of the strips (that is, their size on strike) is variable. With a smaller angle of inclination, stable rocks and other favourable conditions, the strip may be as much as 8-10 metres wide, while in steep beds and with unstable wall rocks, its width decreases to the height of an entry, that is, to 2.5 metres. Mining is done either with a continuous (right side of Fig. 246) or stepped face (left side of the same figure), this depending upon the width of the strip, the hardness of coal and the nature of its cleat. Each coal strip is serviced by two entries—the lower one for the haulage of coal, the upper for the supply of mine-fill. The length of the latter increases progressively, while that of the hauling entry becomes shorter. In the first half of its life each entry is used for the transportation of mine-fill, in the second for that of coal. Since the flow in both instances is in one direction, the entry is driven with a normal gradient.

In order to increase the total footage of working faces the block or panel is made with two wings (Fig. 246) with a moderate size on strike and two or more subdrifts in each level, in which a single strip is worked at a time or, finally, two or several panels are mined simultaneously in each wing.

One major shortcoming of strip mining is the insufficiently large breast front of production faces. This, however, is counterbalanced by the following merits: small volume of development work and its simplicity, safety of mining in working places of limited height and simplicity of ventilation. Coal extraction with strips on strike is most suitable for mining medium-thick and high seams, but with

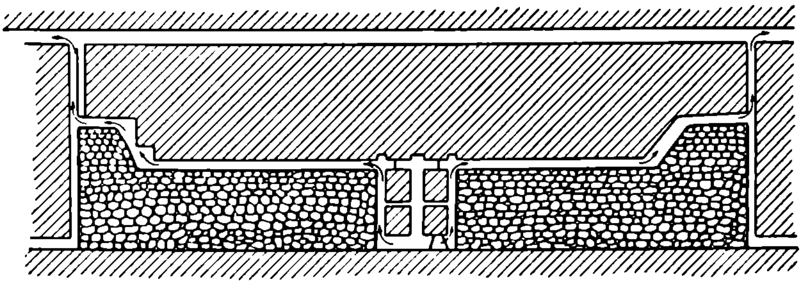


*Fig. 246.* Mining by strips on strike

the thickness not surpassing 3.5-4 metres, with self-igniting coal of diverse hardness and poor wall rocks. It can be practised at various angles of dip, but primarily in steeply pitching beds.

## 4. Strip Mining to the Rise

Fig. 247 is illustrative of this method in combination with shrinkage stoping of coal. The level is divided into two or three subs, each 40-70 metres high (one of these is shown in Fig. 247). Each strip or band of coal *I*, *II*, *III* ... is 6-12 metres wide and is worked out from the bottom up. Broken coal is *shrunk*, that is, left in the mined-out space as provisional fill. But since the volume of broken coal is approximately one-third greater than that of coal in place, part of it is discharged via chutes below the strip and loaded into mine cars (in the haulageway) or into mine cars and onto conveyers (in the subentry), which transport it to a coal dumping chute. The

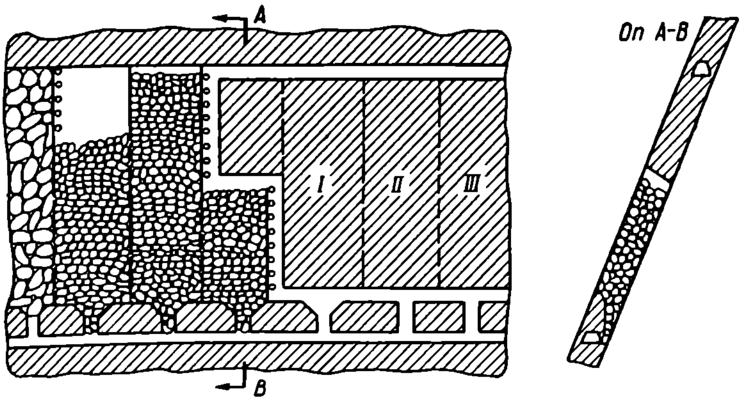



*Fig. 247.* Mining by strips to the rise with the shrinkage-stoping of the coal

discharge of coal is done so as to leave sufficient room between the face breast and broken coal for undisturbed drilling of holes and sorting of loosened coal. To make coal in the shrinkage stope serve as an efficient temporary fill protecting the space near the active face, the portion contained in the strip contiguous to the one mined should be left in place, and stored coal is discharged only from the shrinkage stope of the preceding strip, as shown in Fig. 247.

When coal from the worked-out area has been removed, the wall rocks are made to cave in or the area itself may be packed with waste from the upper entry.

The method described can be applied only in steep seams 2-3.5 metres thick, with hard coal and stable wall rocks. Its practical use is consequently very much restricted.

## 5. Mining with Diagonal Strips

The principal features of the method designed for working sloping and steeply pitching seams are shown in Fig. 248. In each sublevel one diagonal (oblique) strip of coal, inclined at an angle to

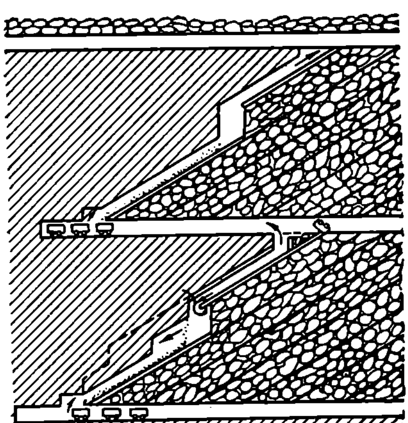


*Fig. 248.* Mining by diagonal strips

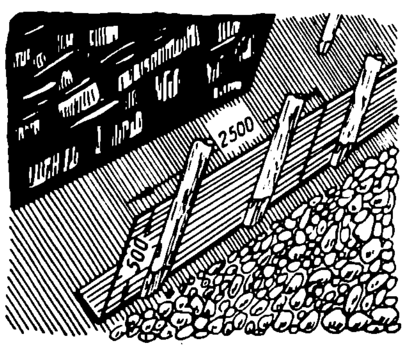



*Fig. 249.* Iron sheet troughs lower coal and fill at the face of a diagonal strip

the strike line sufficient to make coal and fill roll along the face, is worked at a time. Coal and filling materials are lowered along bent iron sheets (Fig. 249) or troughs. Coal is broken in one (upper half of Fig. 248), or two (lower half of Fig. 248) benches. Backfill in the strip is held vertically by a special boarding. The chief shortcoming of this method is low total stope footage. Mining with diagonal strips is applicable to medium-thick and high seams, but not more than 3.5 metres thick, in medium and heavy pitches, with weak wall rocks and self-igniting coal.

## PILLAR MINING METHODS

### 6. Long Pillar Mining on Strike with Caving in Slightly Inclined Beds

When thick beds are worked, use is made of mining methods in which the nature of development is analogous to the sytsems employed in extracting low- and medium-thick coal seams. But thickness lends important peculiarities to these methods which are not inherent in the systems described above in Chapter XIII.

There are modifications of long pillar methods involving caving and filling. In the case of high seams, caving methods, as a rule, present many disadvantages, and therefore we shall describe them briefly, mainly with a view to exposing their negative aspects.

The method of long pillar mining on strike with the subsequent caving of the back has long been used in the Silesian and Dabrowa fields to win coal from slightly inclined beds 4 to 10 metres thick. It is called "Silesian method" for short.

In this case, long pillars are set apart by first mining (Fig. 250). Entries are run in coal, near the bottom of the seam. Long pillars are mined towards the slope, with the upper pillars worked in advance.

Each individual pillar is extracted by cuts to the rise, but the thickness of beds requires a number of special methods.

The cuts or pockets are usually 7-9 metres wide. Coal in each is first drawn by being undercut along the section of the entry corresponding to the cut, with the simultaneous enlargement of the entry up to 5 metres. This is done by firing shots. The enlarged section of the entry with a slashed roof is timbered with caps (Fig. 251) supported by props. The mine timber used here must be large in size and this makes it rather difficult to handle.

The cut or pocket in the pillar is then mined over the entire thickness of the seam. In high faces ladders are used for boring holes with electric hand drills (augers). When mining is in progress, a temporary coal pillar is left on the side facing the goaf (the so-called "stump"). Likewise, the pocket does not reach the goaf near the upper entry, where a portion of coal is also left ("skin"), which, incidentally, is pierced by small "windows", which make it possible to evaluate the thickness of the "skin" and the state of the goaf.

The pocket is supported by headpieces and props (Fig. 251) set up along the strike.

The robbing of the stump is carried on strike, in the direction of the "goaf", starting from top. It would be dangerous to draw all the coal in the stump, and therefore some of it is abandoned on the side nearest the goaf. When the stump is robbed, the headpieces are set along the strike. Fig. 250 shows various phases of pillaring.

When coal has been extracted as much timber as possible is withdrawn from the worked-out space, and the roof is then allowed to cave in. Safety measures in the form of cutting-off supports are taken in the pocket.

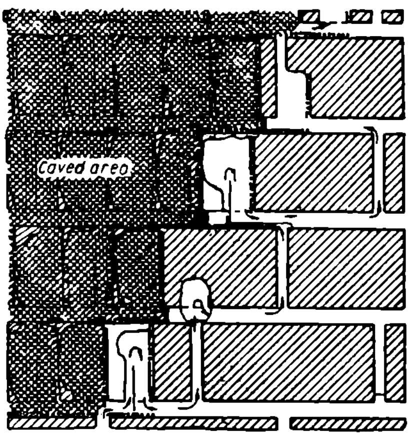



*Fig. 250.* Mining of a thick, slightly inclined bed by the Silesian method

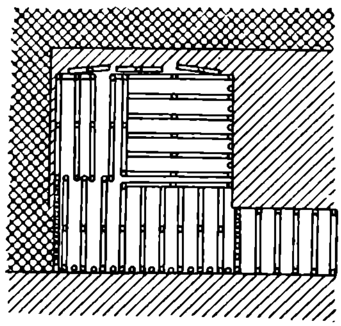


*Fig. 251.* Face support with the Silesian method of mining (plan)

If they are strong, these supports take on rock pressure, thus establishing a distinct "break" line between the falling rocks and the coal remaining in place.

In the case of the Silesian method, the large size and considerable weight of mine timber makes its withdrawal a difficult and hazardous operation which demands experienced workers and special methods and equipment (described in textbooks on mine supports). It is but natural that some of it is abandoned.

After the removal of timber but before caving is proceeded with, a stopping in the form of an organ or battery set, reinforced by struts and braces, is arranged in the entry, near the mined-out pocket. Its purpose is to preclude any possible outbursts of lumpy rocks into the entry during the subsequent collapse of the roof.

If, as the result of the caving of the hanging wall rocks, there is a possibility of water-bearing sands rushing into the worked-out area, the stopping must be made of a filtering type.

Ventilation of working places is depicted in Fig. 250.

The Silesian method has many obvious shortcomings. Since a high seam is mined over its entire thickness without any filling, the collapse and subsidence of hanging wall rocks manifest themselves intensely and, moreover, reach the ground surface. Notwithstanding the presence of cutting-off supports, rock pressure near the goaf is strong and this necessitates abandoning coal pillars. These cause underground fires and to extinguish them the fire-stricken sections have to be isolated by fire seals.

The overall losses of coal caused by the Silesian method are extremely high, reaching 30-40 and more per cent of the total workable reserves. Because of high faces, it is difficult to look after the back of a production place. There is a constant danger of men being injured by falling coal. Handling heavy and long timber is a problem in itself, while its consumption and cost are extremely high.

The mining of thick beds by *cuts* or *pockets*, which is sometimes practised, is essentially analogous to the Silesian method and has the same drawbacks.

The Silesian method is being used in Poland less with each passing year, but in 1954 it still accounted for about 38 per cent of her entire coal output.

## 7. Long Pillar Mining on Strike with Hydraulic Filling in Gently Inclined Seams

The disadvantages of the Silesian method in roof control by caving are eliminated by the use of complete hydraulic filling. An outline sketch of this method is given in Fig. 252. "Extraction of long pillars," writes Prof. A. N. Sidorov in his book *Wet Fill* (1923), "is effected by eight-metre-wide strips to the rise. To form protective chain pillars *a* near entry *OO*, the primary mining of the first 3-5 metres is done by driving headings with narrow faces 2 metres wide and 2 metres high; the face is then broadened to 8 metres and its height is raised to 5 metres. The mining with this enlarged heading ($5 \times 8$ metres) is continued until the latter meets entry $O_1O_1$ and is accompanied by the simultaneous recovery of stump *1* over entry $O_1$. The worked-out space is supported by headpieces at one- or two-metre intervals, each backed by four props. When coal extraction is completed, the worked-out strip is fenced off in its narrow section near entry *OO* by bulkhead *2* and in the upper entry $O_1$ by bulkhead *3*.

"Canvas air-tube *4* is laid on the side of the coal mass along the wall of the mined-out strip to ventilate the next strip. After this, the worked-out strip is packed with wet fill supplied via tubes *5*, laid along the upper entry, through bulkhead *3*. The water first flows down through lower stopping *2* and then via upper stopping *3*.

"The brattice tube has a series of wire rings, spaced at 0.5-0.7 metre, around its circumference; their diameter is about 0.5 metre. When the mining of the next coal strip is begun, vent holes are made in tube *4*, at certain intervals. In this way the air sweeping the working place of the strip is directed to the nearest hole in tube *4* and passes through it to entry $O_1$. Tube *4*, burst or broken, is left in the fill and is considered lost."

At present this method is not used in the U.S.S.R., but its employment in the future is not ruled out, provided there are abundant deposits of sand near a mine and it is suitable for hydraulic fill.

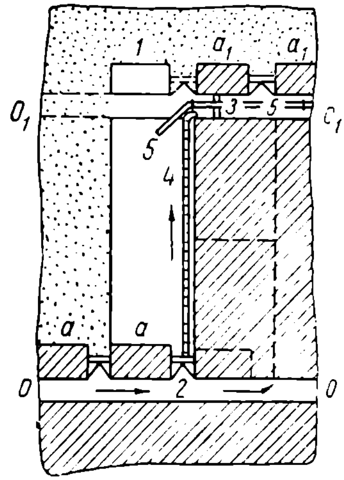



*Fig. 252.* Long pillar mining with hydraulic fill

## 8. Long Pillar Mining on Strike with Caving in Steep Seams

Chapter XIII describes the mining of low seams by long pillars on strike with a stepped or continuous face. A method analogous by the nature of its development work is also applied in the extraction of coal from medium-thick and high seams (not exceeding 4 metres). But the increased thickness of beds lends specific features to the methods of coal breaking, face timbering and roof control.

Let us familiarise ourselves with the variant of this system employed in the Prokopyevsk-Kiselyovsk district of the Kuznetsk coal fields. Intermediary entries divide the level interval of 80-100 metres into two (Fig. 253), three or more sublevels. Since, from the thickness of 2-2.5 metres up timbering of a stepped face in a steep seam becomes extremely difficult, the pillars are worked by continuous faces. Coal is broken by firing shots. Coal, especially in thicker beds, is usually drawn in a narrow (one metre wide) "band", from top to bottom. Broken off, coal moves along the face by gravity. In the subentry it is transported either by conveyers or in mine cars. The face is supported by timber props and caps arranged on strike. In view of the heavy and large-sized props, the timbering of the face is a labour-consuming operation. To keep the overhanging solid mass of coal reliably in place, *roof timber* is set up in the upper section of the coal wall (Fig. 253, detail *C*). With roof control effected by caving in thinner beds, rows of breaker posts on strike are arranged (detail *D*). But in seams more than 2.5 metres thick,

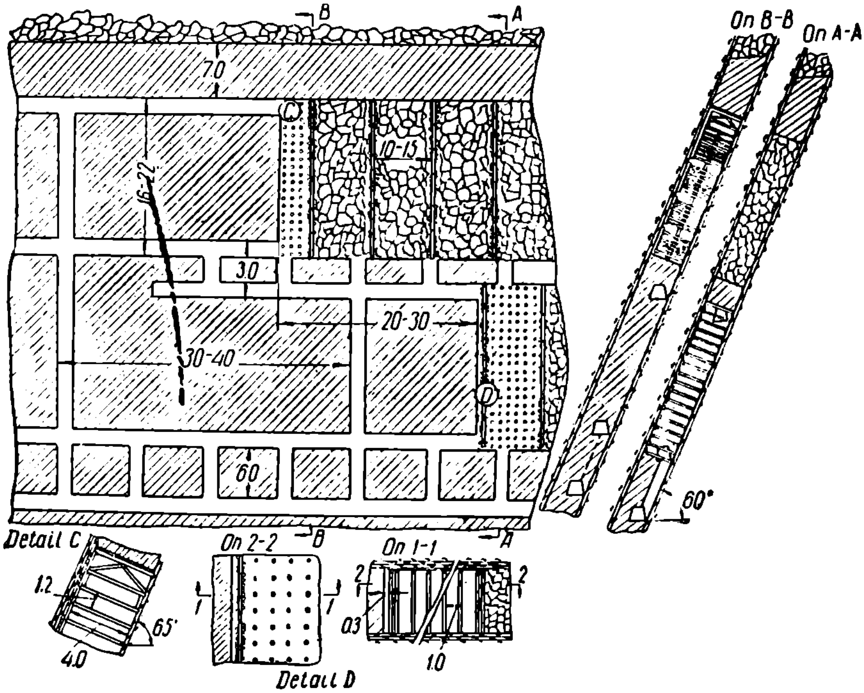



*Fig. 253.* Long pillar mining with caving in steep seams

the general practice was regularly to leave coal pillars on strike, the so-called "rib pillars", since they formed a "break" line for the caving roof. The distance between rib pillars on strike (usually 10-20 metres) corresponds to the "space interval" of the roof break when it caves in spontaneously. The abandonment of rib pillars entails increased losses of coal and should therefore be avoided.

The above method has until recently been widely used in the southern Kuznetsk coal fields in mining steep seams up to 3.5-4 metres thick. Attempts to apply it in thicker beds have caused excessively high coal losses along their thickness. The facemen immediately engaged in breaking coal produced about 13-14 tons per shift, but since 60-80 per cent of labour at the face was tied up in timbering, average productivity per faceman was about three tons. Mine timber consumption per 1,000 tons of coal produced came to 35-40 cu m and more.

Because of all this particular attention should be paid to efforts to introduce powered support (mechanised timbering) in coal walls. So far, however, they have been unsuccessful.

## 9. Long Pillar Mining on Strike with Filling in Steep Seams

An effort to reduce coal losses was made in the mines of the Prokopyevsk district in 1935 with a system of mining similar to the one described above, but with a complete fill. During the war the work was stopped, but today, following the switch-over to the mining of high steep seams in the Kuznetsk coal fields, the method is being applied again in its improved modifications.

1. The delivery of the fill along the subentries tends to create certain inconveniences: coal from the faces of the overlying level and the fill for the underlying level have to be delivered in opposite directions. To eliminate this shortcoming, the following method has been used in recent years (Fig. 254). The subentries are used only for the delivery of the filling material. Coal coming from the production face of the overlying sublevel, the mining of which lags behind the underlying one, is not delivered to the dumping chute on the boundary of a given block, but is passed down along special *coal-dumping* metal or wooden *tubes*, laid in the mined-out space of the subjacent sublevel and then buried under the fill. From the wall to the next coal-dumping tube coal is delivered by a short flight conveyer. With the gravity fill, this is also supplied along the intermediary entry by chain-and-flight conveyers.

2. The method fairly widely used in mining ore bodies (Chapter XX and XXI) is that of working by "subdrifts", or sublevel mining.

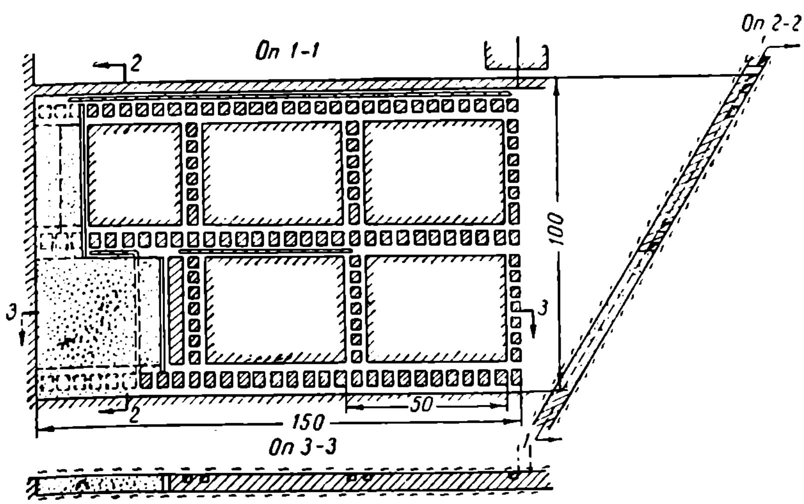



*Fig. 254.* Mining of a steep seam with coal passed down the tubes laid in a solid mass of fill

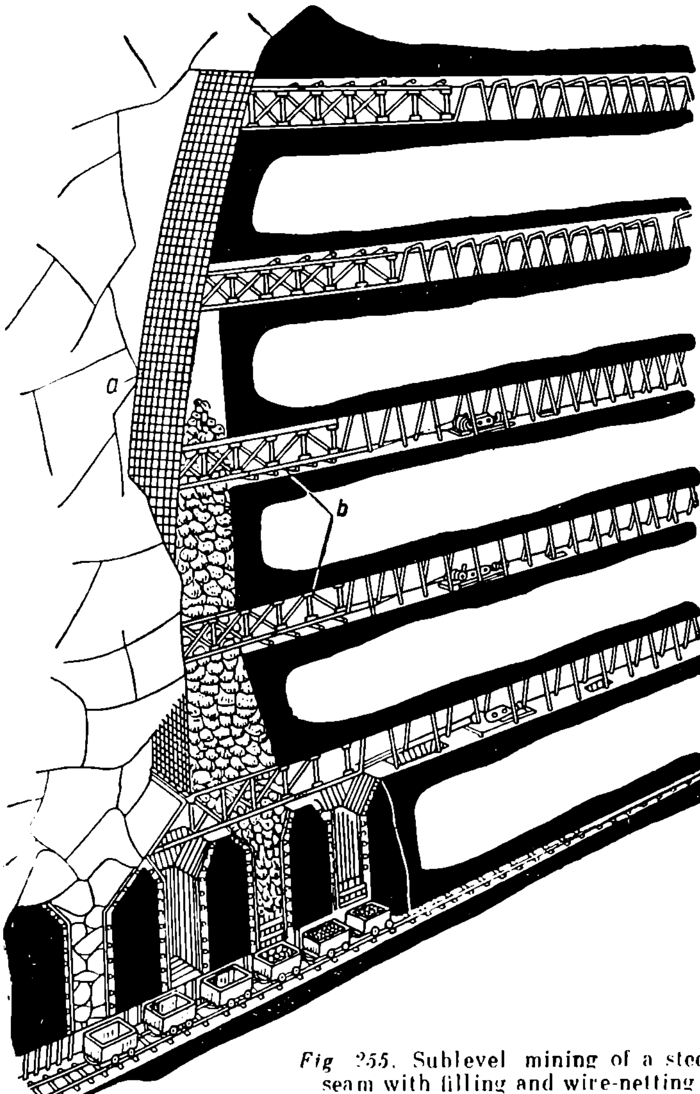



*Fig.* 255. Sublevel mining of a steep seam with filling and wire-netting

This system consists in dividing working sections (blocks in mining of ore bodies) into sublevels of a very restricted height and blasting the mineral from the ends of the subdrifts. The worked-out area is left unsupported and, if the dip is high, the mineral slides down into the lower portion of the level and is then loaded into mine cars in the lower haulageway. Such a method is permissible only in the case of very hard country rocks, and the overall stability of the rocks in the hanging and foot walls is secured by protective chain pillars left near the workings, which causes large coal losses. In the instances where the sublevel mining method is applicable, it presents one major advantage: there is no need to support the stoped-out space, and that is especially important in working highly dipping deposits of considerable thickness.

There have been proposals to apply this method in working steeply dipping coal beds. But since this system entails high losses of coal and the wall rocks in coal seams have proved to be insufficiently firm, nothing has come out of it.

In recent years Engineer P. Kokorin has made tests (so far unsuccessful) of sublevel mining with the aid of a special *wire net* (Fig. 255). This arrangement represents flexible wire net *a* tightly spread on steel frame *b*. As the working face advances, the whole structure, according to the inventor's suggestion, should be pulled by rope lines wound around drums of the hoists set up in the entries. First trials were carried out in mining with caving, but at present a similar device is being designed, for uses involving the filling of the mined-out areas.

The use of such a wire net is effective if the seams are moderately thick (apparently not over 5-6 metres) and the pitch is very high.

## 10. Shield Mining Method in Highly Pitching Seams

By the nature of development shield mining is close to pillar mining (long pillar to the rise) with extraction to the dip. But the feature that particularly distinguishes it from all other systems is the use of a specially designed *shield* for the protection of production faces which is pulled down immediately after their advance. This system, introduced by Prof. N. Chinakal, is an original creation of Soviet mining technology.

The essence of the method is depicted by Fig. 256, which gives one of its typical modifications. Development work is started by raising up the entire level (or sublevel) interval of rise headings (through-cuts), provided with ladder and timber compartments (Fig. 257). If rock pressure is high, these through-cuts have to be supported by cribs. Ventilation in the faces in through-cuts is facilitated by holes preliminarily drilled along their centre line from the lower entry with the aid of a special heavy "break-through" drilling machine (СБМ), designed by A. Mogilevsky (Fig. 258). From bottom up the hole is drilled with a diameter of about 400 mm, and it is subsequently enlarged to 700-800 and sometimes more millimetres. Drilling causes large amounts of dust. To eliminate it, special dust catchers have been designed by the Kuznetsk branch

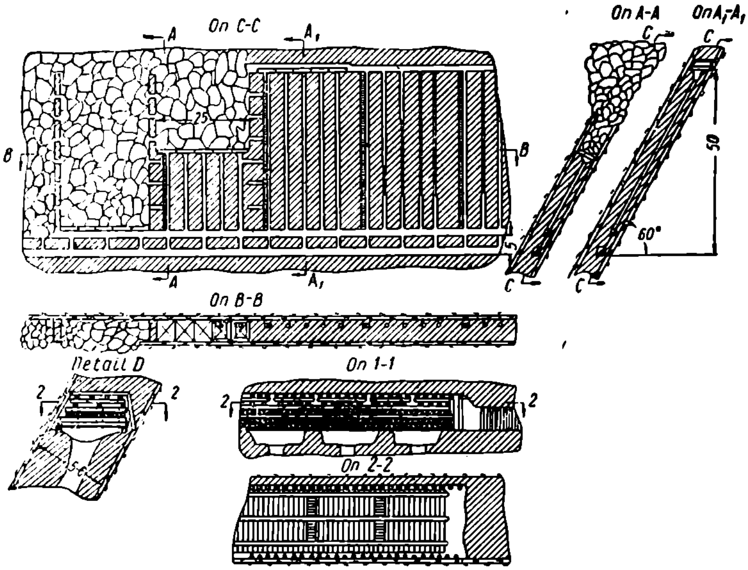



*Fig. 256.* Shield method of mining

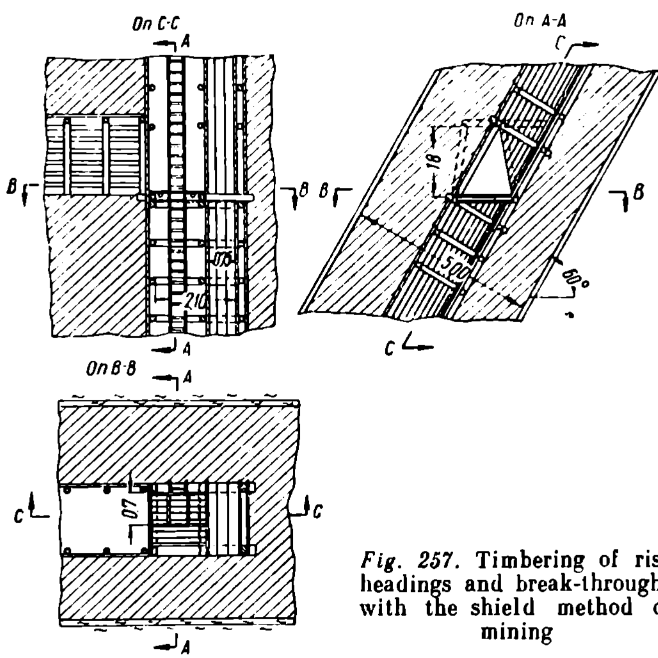



*Fig. 257.* Timbering of rise headings and break-throughs with the shield method of mining

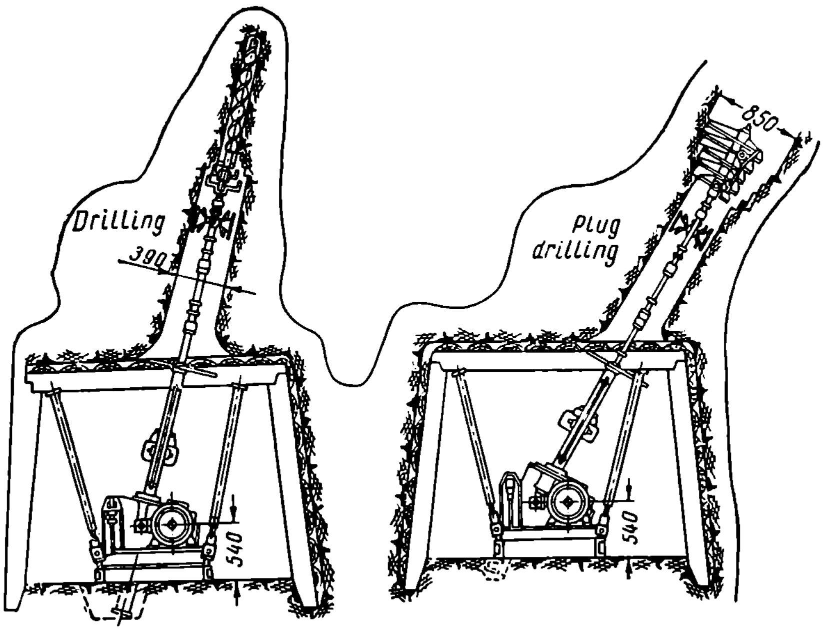



*Fig. 258.* Diagram showing the operation of a heavy-type "break-through" drilling machine

of the State Institute for Designing Coal Equipment on the proposal of B. Vishnevsky. Rise headings are raised every 15-25 metres (at greater intervals in individual instances), and thus delimit a working ("shield") section. The same machine then drills holes, intended for the passage of coal broken down in workings under the shield, every 6 metres on strike.

The shield itself is a light metal structure made of channel and angle iron, serving as a brace for a solid timber flooring. Its width is determined by the thickness of the working seam (Fig. 259). Lengthwise the shield consists of separate, adjoining sections 5-6 metres long. During the actual mining process the shield is moved along the dip. For that reason, a special "assembly" room is built in the upper portion of the level of each shield section to facilitate the erection of the shield (detail *D* in Fig. 256). For the erection of the shield the mouths of the boreholes are enlarged in funnel-like fashion. It is here that the actual breaking of coal is begun with pneumatic hammers and, sometimes, by small charges of explosives. At first, coal is mined chiefly near the foot wall of the seam to make the shield, descending under its own weight, assume its regular working position (Fig. 259). After this it starts gradually to come down, its position all the time being normal to the bottom and back of the bed. The forward movement of the shield is regulated by the sequence of coal breaking—now near the back and now close to the bottom of the seam.

The shield, therefore, is a controlled movable structure capable of protecting the active stope space and pushing forward with the advance of a working face. It should be lowered to the very bottom portion of the level, where it is dismantled and transferred to a new working section. However, experience shows that some elements of the shield are gradually disarrayed in descent and when the structure covered by a layer of caved rocks is finally lowered, it is no longer considered worth while dismantling it.

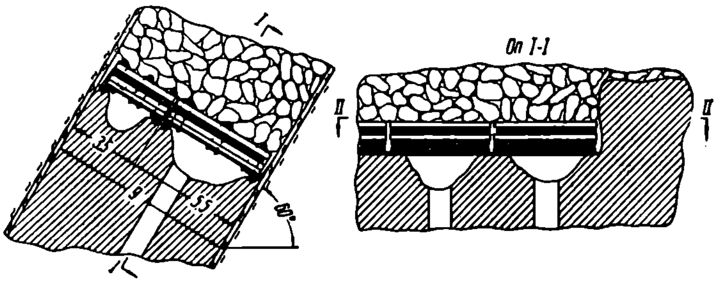


*Fig. 259.* Shield in working position

The total length of the shield should be somewhat less than the size of the section on strike and, therefore, as the faces advance down the dip, coal pillars form on their side—between the active and worked-out sections or between the active section and the rise heading.

As shown by Fig. 257, in the case of shield method of mining, production faces have two exits—via the two rise headings delimiting the working section. The ventilating current follows the same course.

The shield in question is distinguished by the fact that the width of its sections is equal to the thickness of a seam. A shield of this design is usually called single-type. Single shields are employed in beds up to 6-7 metres in thickness, and those of reinforced type in beds 8-10 metres thick. For working thicker beds, it has been suggested to use double- and even triple-type shields, whose sections along the thickness of the bed are made up of two or three separate parts. But control of the descent of double and triple shields has proved an extremely difficult affair.

Latterly steps have been made to introduce a newly designed type of shield, the so-called *flexible nonsectionalised shield.* It has but one row of timbers (dead or counter floor), each of which is sawn on both opposite sides and all joined together by channel iron and bolts. The timbers are arranged perpendicularly to the bottom and the roof of the seam. The top face of the shield is covered with a metal net. This flexible shield is not divided into sections. As field experience shows, a flexible shield is a supple structure, and that makes it easier to control and handle. The metal wire netting prevents rock pieces from falling through the interstices between the timbers. Unlike the ordinary shield, the flexible one greatly reduces timber consumption.

The shield method of mining presents many important advantages: efficient and safe work in production faces and coal is delivered to the lower entry by gravity, requiring no lateral haulage.

On the other hand, the method has a number of important drawbacks. It causes considerable losses of coal. These are due to the abandonment of coal pillars referred to above and to the waste of coal over the entire thickness of seams. The fact is that the working thickness of a bed (that is, the height of the mined-out space) is predetermined by the design of the shield and the increase in the thickness of the bed is liable to result in "crusts" of coal being left near the bottom and the back of the bed. Moreover, it frequently happens that the shield comes into disarray before it reaches the lower portion of the level ("danger section") and the affected part must be extracted by the room-mining method (Section 11), which involves extremely high losses of coal. Since the shields cannot be dismantled, consumption of timber is as much as 20-30 cu m per 1,000 tons of coal produced. On top of all that, some 1.5 kg of metal is lost per each ton of coal mined. The extraction of coal and the descent of the shield are followed by the dislocation and caving of wall rocks in the stoped-out areas, among which there may be some abandoned coal, and this can lead to the outbreak of underground fires.

The coal seams most suitable for the effective use of the shield method of mining are those 4-7 metres thick, dipping at an angle of not less than 55-60°, with regular occurrence and hard coal. In certain instances, however, this system may also be employed in working thicker and thinner seams starting with three metres.

In recent years the shield method of mining has accounted for about 40 per cent of the coal produced by the Prokopyevsk-Kiselyovsk district of the Kuznetsk coal fields. Each shield section yields around 6,000-8,000 tons of coal per month, and considerably more in some cases. Output per man per shift in the section is as high as 6-7 tons.

One major disadvantage of the shield method of mining is the appreciable losses of coal. This is aggravated by the fact that the use of the system involves the caving of the overlying rocks, which is fraught with the danger of underground fires caused by the spontaneous ignition of coal. For this reason Prof. N. Chinakal and a number of other specialists are working to improve the method by combining it with filling. The solution of this problem is rendered difficult chiefly by the installation of a reliable support for the upper portion of the mined-out area, which is necessary if the filling material is to be supplied unhindered. Another means of reducing the waste of coal when the shield method of mining is employed is envisaged in the suggestion regarding the preliminary recovery of coal pillars and their replacement by pillars of barren rocks cemented by cheap materials (ground steel mill slags, for example). This second method is still in the stage of research and planning.

The mining of the uppermost level is simple and original with this method with filling. When the outcrops of the seam are covered with overburden the lowering of the shield is accompanied by the caving of the superimposed rocks which gradually fill the worked-out space. This causes holes and depressions on the surface which are filled with mantle rocks by bulldozers.

## 11. Room Mining

The room methods of mining are considered obsolete and inefficient for working coal beds.

In the 1930's room mining was applied in the gently inclined Verkhnaya Marianna seam of the Karaganda coal basin. It was about 8 metres thick and of a rather complex structure.

This method of mining was attended by excessive coal losses (40-45 per cent) and the appearance of huge holes on the ground surface. This prompted the discontinuation of this method and the adoption instead of that of rill mining (inclined slicing) (Section 19). In 1931-36 room mining was widely used in the Kuznetsk coal fields in working thick steeply pitching beds.

The distinguishing points of the system are as follows (Fig. 260). From the main entry driven in coal closer to the bottom of the seam, rise headings in the form of twin openings are raised every 10-12 metres and are later connected by crosscuts. Discharge chutes for loading coal into mine cars are installed at the bottom of the rise headings in the entry. Over the cap pillar of the entry the rise headings are connected by a break-through and then enlarged funnel-like to form a bottom for the future room or chamber. Coal is blasted in the roof of the room. Thus loosened, coal is not drawn off completely from the chamber but only in amounts corresponding to its expansion on breaking. The shrinkage method is meant to serve a double purpose—broken coal is left in the face as a floor on which miners stand while working the back and as temporary filling for the room. Communication between the room face and other workings is only

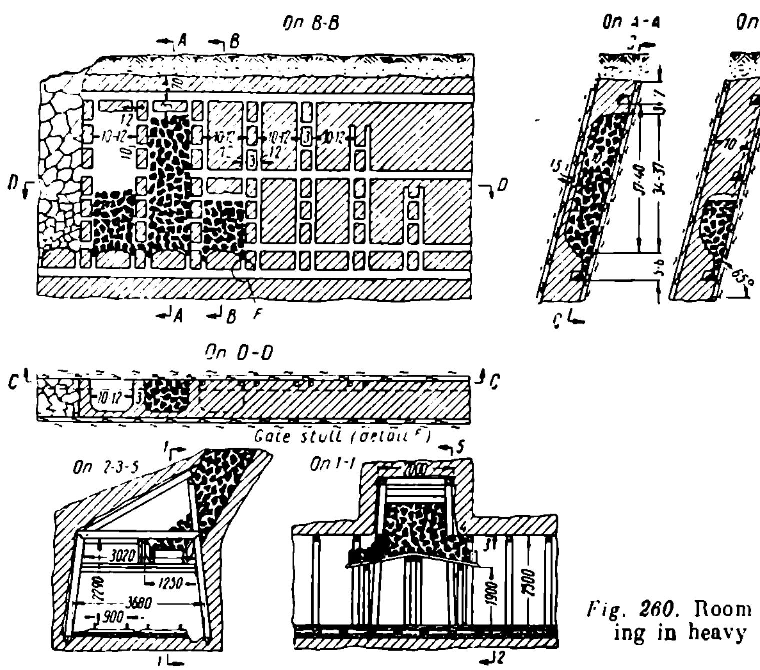


*Fig. 260.* Room mining in heavy dip

possible via the rise headings. These also serve as routes for ventilating air currents.

Coal thus stored is ultimately drawn off when the room has been worked to the level of the upper entry. Fig. 260 depicts three rooms: the one on the left is in the drawing-off stage, the central—coal has been broken and its shrinkage completed, the one on the right is in the breaking and shrinkage phase.

When this method is used, interchamber or rib pillars of about 3 metres in width are lost completely and irretrievably. In the process of drawing off stored coal, the wall rocks and those overlying the back of the room, as well as the ones adjacent to the pillar sides, cave in and begin sliding down. Sometimes these phenomena become violent before the drawing-off operations have been completed ("spontaneous collapse of the room").

Room mining practised in this way yields high coal output per man per shift (6-8 and more tons), but it also has some major disadvantages:

excessive waste of coal—about 40 per cent in beds up to 5 metres thick, and not less than 40-50 per cent in thicker seams;

consequently, increased danger of underground fires on account of the spontaneous ignition of coal;

impossibility of separating barren rock partings from coal during the stoping process and this, in seams with abundant gangue intercalations, increases the ash content of coal;

dislocation and movement of ground over the rooms cause heavy damage to the surface;

work in weak and fractured coal is far from being safe in large exposed areas.

During the Great Patriotic War this method accounted for up to 15 per cent of the entire amount produced in the southern part of the Kuznetsk coal fields, but today it is prohibited.

The system under discussion should not be confused with the *room-and-pillar* method of mining, which is distinguished by the regular, though sometimes partial, robbing of interchamber pillars, this appreciably reducing the overall losses of the mineral. But since the application of the latter is characteristic of mining medium-thick seams, it is discussed in Chapter XIV, Section 3.

## B. SLICE MINING OF THICK BEDS

### 12. Division of a Thick Bed into Slices

The thickness of each slice should be such as to make its extraction easy and convenient. More often it is 2-3 metres, less frequently—up to 4 metres.

One can imagine many ways in which a thick bed can be divided into separate slices by being cut in various directions with parallel planes, spaced at distances equal to the actual thickness of the slice. In practice, however, it is only the following four methods that are known to be in actual use:

1) *inclined slicing*, with slices running parallel to the plane of the bed (Fig. 261*a*);

2) *diagonal slicing* or rill cut mining (Fig. 261*b*). Here the position of slices in space is also inclined. However, they do not extend on strike, but across it, between the bottom and the back of the seam;

3) *horizontal slicing* (Fig. 261*c*), that is, working with slices limited by horizontal planes;

4) the so-called "*cross-inclined*" or "*transversely inclined*" slicing (Fig. 261*d*), where slices are inclined at an angle of about 30° between the hanging and the foot walls of the seam. Consequently, with a bed dipping at around 60° such a "cross- or transversely inclined" slice lies normally with regard to the planes of bedding.

Separation of a bed into *vertical slices*, limited by vertical planes running across the strike, is no longer practised.

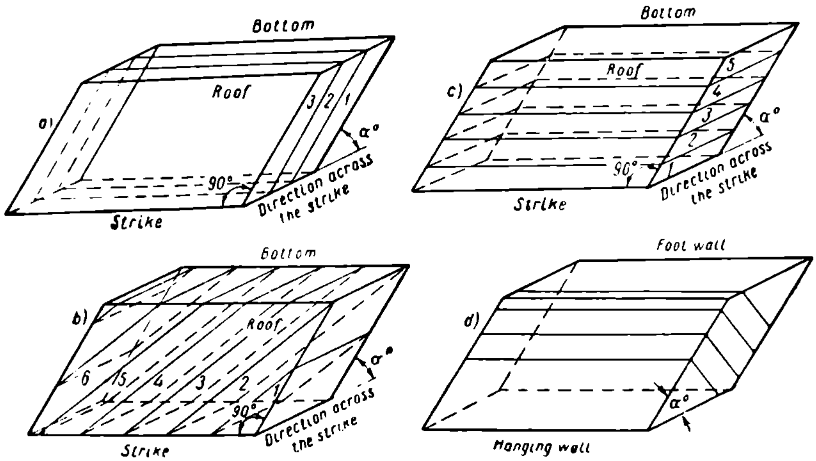



*Fig. 261.* An outline of slice positions

a—inclined slices; b—diagonal slices; c—horizontal slices; d—transverse (cross) inclined slices

INCLINED SLICING

## 13. Division of a Bed into Slices and Their Thickness

Inclined slices into which a thick bed is divided for mining purposes may be in the form of individual benches, if they have a permanent structure, suitable thickness (2-4 metres) and are split by interlayers of barren rock, or else a seam of a homogeneous structure is artificially separated into several inclined or rill slices.

The thickness of slices depends on the following considerations. To provide sufficient free room for men to stand in the face this thickness should not be below 2 metres. In inclined slices a man can stand freely even if the slice is somewhat thinner. But this can be allowed only when contiguous slices are confined to the separate benches of a complex seam, split by partings of gangue.

The thicker the slice the smaller the number of slices into which a given bed is divided, and this presents a sizable advantage in terms of reduced volume of development work. On the other hand, a slice of considerable thickness entails many inconveniences:

a) in high faces breaking of coal in the upper portion of the slice becomes rather difficult;

b) the greater the thickness of a slice the more difficult it is to keep a constant watch over the condition of the roof and the upper portion of the face to ensure proper safety of mining;

c) high faces demand the use of long, that is, heavy and hard-to-handle timber or metal posts;

d) in the modifications of the method involving caving, roof control becomes a complicated matter;

e) in working with filling it is also difficult to bring the pack up under the roof of a given slice. This was a decisive factor in restricting the thickness of the slice when filling was done by hand, but its importance has diminished greatly with the advent of power stowing.

The sum total of the factors enumerated above prompts to restrict the thickness of slices to 2-3 metres, the upper limit of 4 metres being hitherto sometimes allowed only in instances of hydraulic filling. The utilisation of stowing machines or pneumatic fill also tends somewhat to increase the thickness of slices.

The above-mentioned considerations not only apply to the selection of the thickness of inclined slices, but are also valid for the comparison of conditions in which seams of different thickness are extracted. They explain the fact, paradoxical as it might seem at first, that the maximum output per faceman per shift is generally achieved not in thick beds, but in those of medium thickness (around 1.8-2 metres).

## 14. Order of Slice Extraction

The order of slicing may be *ascending* (that is, from the bottom to the roof in sequence 1, 2, 3 in Fig. 262*a*), or *descending* (from the roof to the bottom in Fig. 262*d*). Each has its advantages and shortcomings in different conditions.

Let us first consider the features peculiar in this respect to mining with caving and backfilling.

Let us assume that a certain thick, slightly inclined seam has to be divided into three inclined slices (Fig. 262*a*) to be worked. With the ascending order the first to be extracted will be the lowest slice (Fig. 262*b*). Its floor is the bottom of the seam and roof--the coal of the overlying second slice. Generally speaking, the first slice will be extracted in conditions similar to those pertaining to mining individual seams of medium thickness. Therefore, the methods of mining and modes of extraction already described can be applied here too.

But it is obvious that ascending slicing excludes *caving*, for after the extraction of the first lower slice and its subsequent caving coal in the superjacent slices would be so fractured (Fig. 262*c*) that it would be impossible to work these slices.* Besides, self-combustion may cause underground fires in the solid mass of fractured coal.

* True, there have been rare cases when, in ascending slicing without backfilling, coal and rocks in the first slice do not cave in, but tend to come down grad-

If, in mining with caving, the inclined slices are extracted successively downward (Fig. 262*d*), the working of the uppermost slice does not present any special difficulty, since the back of the seam will be its roof and the coal of the subjacent slice its floor. But when this slice has been extracted and its roof caved, the conditions for

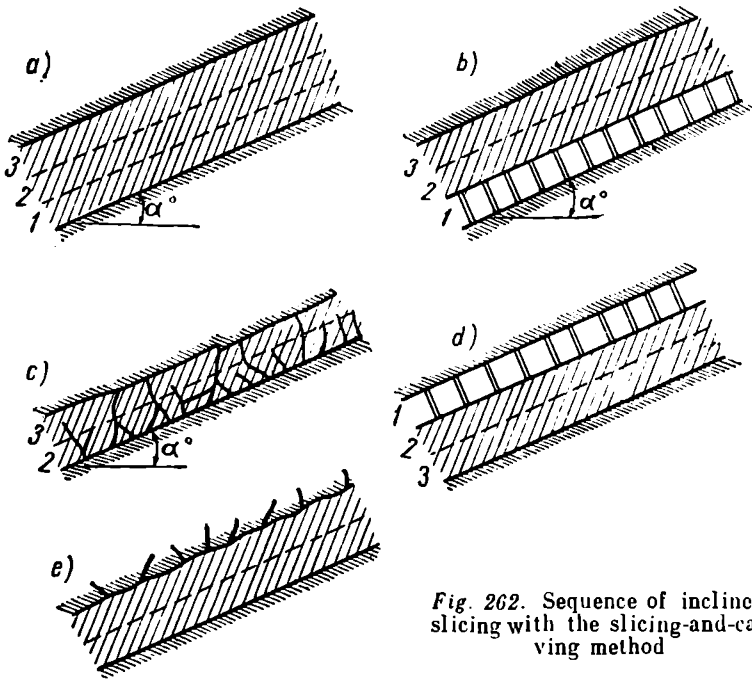



*Fig. 262.* Sequence of inclined slicing with the slicing-and-caving method

mining the second slice from the top will be quite different. Its floor will include coal from the next slice, but the roof will contain caved rocks belonging to the back of the seam (Fig. 262*e*). The extraction of a slice under such a roof is usually difficult. However, the degree to which these difficulties make themselves felt depends, in a great measure, on the properties of the rocks in the roof of the seam, on the availability, properties and thickness of gangue intercalations between the slices, the angle of dip of the seam, the percentage of coal recovery in the superimposing bed and the time interval between the actual mining of the subjacent and overlying slices. The

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ually and their continuity remains intact. This gradual subsidence is facilitated by the physical properties of the rocks and the availability in the goaf of pliable cribs built of thin sticks.

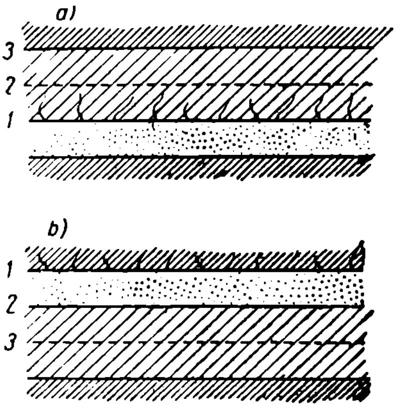



Fig. 263. Sequence of inclined slicing with the slicing-and-filling method

influence exercised by all these factors is discussed below.

With complete *filling* both the ascending and the descending slicing is possible.

When the lowest slice of a thick bed is extracted and properly filled (Fig. 263*a*), its fill serves as a floor for mining the second slice, whose roof is made of coal in the third slice. The other superjacent slices can then be worked out successively in the same way. But this depends on the quality of the mine-fill. The less its shrinkage the lesser the degree of disturbed continuity in coal in the overlying slices and the more normal the conditions for their extraction.

The contraction of the fill is minimal in the case of hydraulic and pneumatic stowing.

The thicker the bed and, consequently, the larger the number of slices it is divided into, the higher the absolute value of the shrinkage of the fill block. This explains the paramount importance of having a proper, tight filling in thicker seams.

The abundance of fissures in coal, caused by its settlement following contraction, may greatly complicate its winning. On the other hand, slight cracks, unavoidable even in a fill block of very high quality, may conversely be a positive factor and facilitate the breaking of coal.

However, in seams subject to spontaneous combustion, fissures in coal present a considerable danger, for, allowing air into the solid mass of coal, they may lead to fires caused by self-ignition.

The reverse, descending order of mining is depicted in Fig. 263*b*. Here the uppermost slice is extracted in conditions identical to those in mining a medium-thick seam, but the roof of the next, subjacent slice will contain the mass of filling materials. Since these are loose, they may, despite the huge pressure exerted by the subsiding rocks, retain their quality of looseness and will form, generally speaking, a very poor roof. For this reason the extraction of inclined slices with filling from top downward is practised very seldom. The main reason which sometimes motivates the use of this order is the susceptibility of coal to spontaneous combustion, a feature which prevents the application of ascending slicing on account of the hazards of underground fires that may arise following the fracture of a coal block. The fitness of a solid mass of mine-fill as a roof in the working slice is determined by the quality of the filling materials used for this purpose.

In ascending slicing, the slice has to be stowed with fill, chiefly for the purpose of preserving the continuity of coal in overlying slices and also in order to have an artificial floor in the process of extracting it. But in working the last, uppermost slice, these considerations obviously drop off, for it can be mined with caving even if all the underlying slices have been extracted with filling.

Hence, here we have to deal with the *combined* mining of inclined slices based on *filling* and caving. This method cuts the cost of filling materials considerably.

The problems of extraction with complete filling or roof caving have been considered exclusively from the viewpoint of the influence these methods exercise on the merits and disadvantages inherent in this or that order of mining mineral beds. But these methods cause movements of varying degrees in the superjacent ground, and are apt *to manifest themselves in diverse ways on the ground surface.* Mining thick beds with roof caving causes a considerable movement of the ground, leading to the formation of hollows, conical depressions, gaping rents, etc., on the surface. There are cases when, to protect the surface and ground structures from any possible damage, one has to renounce caving methods. In this connection it may be noted that even a good filling material merely attenuates the subsidence and movement of the ground, but does not eliminate them completely.

## 15. Effects of the Angle of Pitch and Wall Rock Properties

Let us see how much the applicability of inclined slicing depends on the *pitch of a seam.*

In methods involving *caving* this system is unsuitable for the exploitation of steep beds. Caved rocks in the roof of a slice, barring the uppermost, would constantly threaten to slide and collapse. An attempt to win thick, steeply pitching seams by inclined slices with caving in a descending order was made in 1930 at the Prokopyevsk mine of the Kuznetsk coal fields and, as should have been expected, the result was definitely negative. For this reason, inclined slicing with caving of thick seams without gangue interlayers, whose thickness and position could make them serve as slice boundaries, should not be resorted to in beds pitching at an angle exceeding 30-35°. If there are gangue partings separating the slices, this system can be adopted even if the angle of dip is somewhat greater.

The U.S.S.R. has so far accumulated limited experience in working thick steep seams with *filling.* But there are already two trends manifesting themselves: descending slicing in the Chelyabinsk coal fields (Section 21) and ascending slicing in the Kuznetsk coal basin (Section 20). This difference is presumably due to the properties of wall rocks and filling materials. In the Chelyabinsk coal fields wall rocks are plastic while the filling materials of argillaceous rocks tend to become compressed and compact, and thus form a satisfactory roof for an inclined slice. The conditions prevailing in the Kuznetsk basin call for filling materials that are less coherent and compressive, and although this makes it possible to have a filled slice in the bottom of the next slice to be mined at a given moment, it does not permit us reliably to hold the fill in the roof of the slice in its place, a feature that is indispensable with descending slicing.

When contiguous inclined slices are mined from top to bottom with caving, it is the *rocks of the back* of the seam, earlier caved into the worked-out space of the upper slice, that form the roof of the next slice. The properties of this artificial roof largely depend on the physical and mechanical features of the caved rocks, the time interval between the extraction of the upper and the lower slices, and the percentage of coal actually recovered from the top slice.

To facilitate the mining of a slice under the caved area, the important thing is that the caved rocks should be a more or less compact mass. If the back of a seam contains hard rocks (sandstone, hard slates), they form a very good roof for extracting the uppermost slice. But after the caving, these rocks are broken by numerous fractures into large individual lumps, and it may prove extremely difficult to hold them in the roof while the second slice is worked. It is because of their hardness that these rocks do not become compressed and so form a weighty and collapsible roof for the underlying slice.

If, on the other hand, the back of a thick bed is made of soft, plastic shists or slates, which require stronger and more elaborate support when the uppermost slice is mined, these shists become well compressed and compact when they collapse under the immense pressure of the subsiding rocks and may prove to be a quite satisfactory roof for working the second and consecutive slices.

Thus, to assure successful extraction of an inclined slice under the superjacent one that has caved in, it is more propicious to have the back of the seam consisting of plastic argillaceous shists than of hard slates and, especially, sandstone.

## 16. The Effect of the Time Element and Percentage of Coal Recovery

The above-described compaction and compression of rocks do not occur at once, since their caving and settlement take *some time.* The subsidence of rocks at any given site of a mined-out area largely depends on the position of the production face with respect to the site. A zigzag outline of the production face front helps keep the roof in place near the face area. Roof rocks break completely only when the faces have been pushed forward to a certain distance. The caving process at first does not affect the whole mass of undermined rocks, but their individual layers or groups tend to come down one after another. Incidentally, the latter phenomenon is more characteristic of hard stable rocks, while in soft plastic shists caving and subsidence rapidly extend vertically affecting thick strata. Finally, some time is necessary for the caved rocks to become compressed after maximal pressure has already set in.

As the result of this, the subjacent slice under the caved rocks of any given inclined slice can be mined only after a certain period, usually from several weeks to several months.

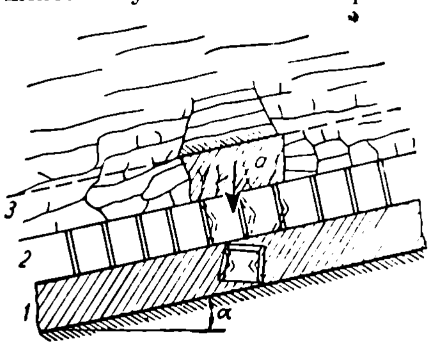


*Fig. 264.* Adverse effect produced on the roof by a layer left in the top slice of coal

The property of rocks caved after the mining of any given slice, viewed in their capacity as a roof of the slice underlying it, is very much influenced by the *percentage of recovery* of the useful mineral in a given slice and the manner in which the abandoned coal is distributed in it. If the extraction of the mineral entails losses in the shape of "stumps", "pillars", "skin", etc., these impede the complete subsidence and compaction of rocks settling over the mined-out areas. Moreover, these abandoned portions of the mineral produce an extremely unfavourable effect on the stability of the roof in the subjacent slice, transmitting *concentrated* rock pressure not only to the timbering of the slice that is mined, but even to the development workings driven in the next slice (Fig. 264). As revealed, for instance, by the experience gained in the Chelyabinsk coal fields, the change-over from inclined slicing by stub entries, open-end mining, etc., all of which systematically caused numerous losses of small coal pillars, to working by continuous faces, where the losses of coal are much less frequent, has markedly improved conditions for the mining of subjacent slices (as regards the quality of their roof). In continuous mining, compaction (compression) of rocks takes less time and is more complete and it is, therefore, possible to start working a subjacent slice under the mined-out space of the one lying over it much earlier.

A high percentage of coal recovery is also a factor of paramount importance for reducing the hazards of underground fires through the *spontaneous combustion* of coal. Crushed and loosened, the small pillars of coal abandoned in the worked-out areas become self-igniting. The numerous fractures and voids forming as the result of the "abandonment" of a portion of coal in working faces and the consequent incomplete caving and subsidence of the capping facilitate the access of air. And conversely, complete extraction of coal minimises the danger of fire, and that not only because there is no or little coal left in the mined-out area but also because the process of the caving and subsidence of rocks ends quickly, thus isolating the worked-out area from the inflow of air (it should be borne in mind that it is not only coal that is subject to spontaneous combustion but also carbonaceous wall rocks).

As far as the high percentage of coal recovery is concerned, inclined slicing with continuous faces (longwalls) is considerably superior to the pillar-and-bord or open-end methods of mining.

## 17. Effects of Interslice Gangue Partings and Coal Strata

It has been earlier mentioned that with an appropriate structure of seams contiguous inclined slices may be divided by intercalations of gangue. The availability of such intercalations greatly facilitates the extraction of subjacent slices. When the rocks cave in in the process of the extraction of the overlying slice, the parting remains intact (though in some instances its original position and properties may be deranged somewhat by the "bulging" and penetration of mine timber into its mass) and thus separates the superincumbent broken rocks from the stoping area of the given slice. The thicker the parting and the harder the rocks it is made of, the more significant the part it plays.

When there is no gangue parting, the bottom of the mined slice is sometimes laid out with slabs or boards. This *flooring* permits it better to hold the broken rocks in the roof of the slice. During the mining of the subjacent slice props can be blocked against the slabs (or round timbers) of the flooring. The size and position of slabs are chosen to conform to the mode of timbering adopted for the underlying slice. This method is known as *temporary* or *preliminary* timbering (for detailed description see Section 19). This temporary timbering can be employed in slicing-and-filling, as well as in descending slicing. In similar conditions, prior to filling the mined-out area, the bottom of a slice is sometimes covered with a layer of clay about 0.5 metre thick. The immense pressure exerted by the settling rocks compresses this clay and thus helps hold the fill in the roof over the active stope area.

In recent years there has been a growing tendency to use *metal wire netting* instead of flooring, or together with it.

In view of the difficulties encountered in mining slices whose roofs contain caved rocks, it is the practice sometimes to leave an *interslice stratum* of coal about 0.2-0.5 metre and more thick, even when working undiluted seams. This facilitates stoping operations and makes it possible to begin them earlier, without waiting for the caved rocks of the upper slice to compress and compact. But this practice is absolutely inadmissible from the viewpoint of coal losses and fire hazards.

### 18. Development Order

In inclined slicing, the order adopted for driving development openings depends primarily on the sequence of slice extraction.

Since, according to the very principle of the method, each slice is regarded as a medium-thick seam, it would appear at first glance that all development workings should be consecutively re-run in every one of the slices extracted. That, however, would require driving too many development openings and their subsequent maintenance. Hence the natural tendency to make most, or at least some of them, serve two or several inclined slices.

As a rule, level haulageways and airways are made to serve all the slices and are usually driven in coal near the foot wall. With heavy rock pressure, they can be made as lateral entries running in foot wall rocks at some distance from the seam.

To connect the level entries with inclined slices, *crosscuts* are driven over distances predetermined by the method adopted for the development of each individual slice. In the case of slicing in flat or very gently sloping beds, it is necessary to raise a blind shaft instead of crosscuts.

The simplest thing is to run independent development openings within the range of a given slice in accordance with the method of mining used for its extraction. After the slice has been worked and subsequently filled or caved and after a lapse of an appropriate period of time (see above), the miners set to work the next slice, which is developed independently and in good time, except for the level entries. One advantage of this order is the independent position of development openings in adjacent slices, which facilitates the planning of work.

By making some openings serve two or several slices, it is possible—though it somewhat complicates the pattern of work—to decrease the number of development openings and, consequently, the cost of their maintenance.

Each slice is extracted by one of the mining methods applied in working medium-thick seams. As a rule, this is one of the modifications of long pillar mining on strike with the recovery of pillars by continuous faces or, with a small level interval, the longwall system.

Sections 19-22 cite typical examples of mining thick coal beds by inclined slices in different conditions.

## 19. Inclined Slicing-and-Caving in Gently Sloping Beds

Brown-coal (lignite) occurrences in the Chelyabinsk coal basin belong in the Jurassic age category. The coal beds, usually separated by numerous intercalations, occur amid relatively weak wall rocks, represented by soft plastic clay shales. These deposits include seams

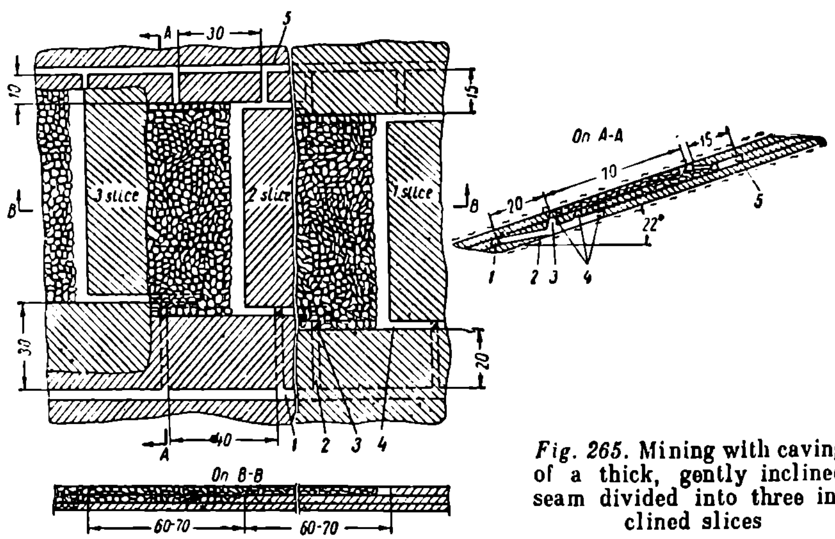



*Fig. 265.* Mining with caving of a thick, gently inclined seam divided into three inclined slices

of varying thickness. The angles of dip range from horizontal to very steep, with slightly inclined and sloping pitches predominating.

In this area inclined slicing is very widely used. In slightly inclined and sloping beds slices are worked with caving in descending order. Each is extracted by continuous mechanised faces, this facilitating the mining of the subjacent slices (see above). When the inclined height of the level interval is not too great (not exceeding approximately 150 metres) the predominant method is that of longwall mining. When the vertical distance between levels is greater, they are divided into two sublevels (stepped longwalls), with the blocking out of long pillar on strike.

Fig. 265 is illustrative of the method employed in working a slightly inclined seam of 6.5 to 8.5 metres in thickness. To draw it, the seam is divided into three inclined slices, each 2.2-2.7 metres high. These are extracted by walls whose length generally ranges between 70 and 150 metres. Lower haulage entry *1* and upper airway *5*, common for all the three slices, are driven in the central portion of the seam which, however, is not recommended, for it is better to run main entries near the foot wall of the bed, or in the country rocks, as stone workings. Each slice has its own independent entries *4*. From the upper slice entry coal is brought along short dumping chute *3* to rise heading *2* to be subsequently loaded into mine cars spotted on the haulageway. In the walls the coal is transported by chain-and-flight conveyers.

Since work is done by descending slicing-and-caving, the walls in the overlying slices are run about 60-70 metres ahead, this corresponding to about three months of actual mining.

Coal is drawn by a mine combine or with preliminary undercutting effected by a coal cutter. The space interval between two consecutive breakings or cavings of the roof equals one cut. In recent years metal posts (Fig. 266) have been introduced in the Chelyabinsk coal fields for the support of coal walls, since, because of the low stability of the roof, consumption of mine timber is excessively high—up to 60 cu m per 1,000 tons of coal produced.

To create propicious conditions for the subsequent extraction of a subjacent slice, a *preliminary support* is set up on the floor of the upper slice. The arrangement is shown in Fig. 267. This support consists of sills (round or sawn timbers) *a*, laid upright of the breast of the coal face. The sills are 2.66 metres long arranged at intervals of 1-0.8 metre. Boards *b* form the flooring of the sills (the boards are also 2.66 metres long and 3 cm thick). Each board rests on three sills and their ends overlap one another. To make the board flooring contiguous to coal, the sills are placed in recesses (ditches) cut out in coal. To prevent rock pieces from falling through the interstices between boards during the extraction of the second slice the flooring is covered by short boards *c*. Experience shows that such flooring greatly facilitates mining operations in the lower inclined slice. As the drawing of coal in this slice progresses, it suffices to block ordinary props against the gradually denuding sills. Furthermore, the flooring prevents coal from being contaminated by gangue. Roof control in the lower slice is effected by caving caused by the shifting of cribbed supports.

Thick, slightly inclined coal seams are also mined by the inclined slicing-and-caving method in the Kuznetsk (Tom-Usinsk district), Karaganda, Cheremkhovo and certain other coal basins.

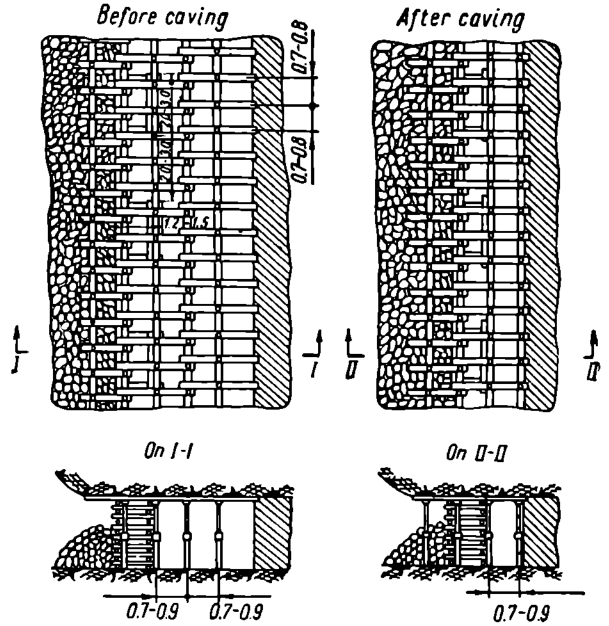



*Fig. 266* Details illustrating production face support with metal posts and timber cribs

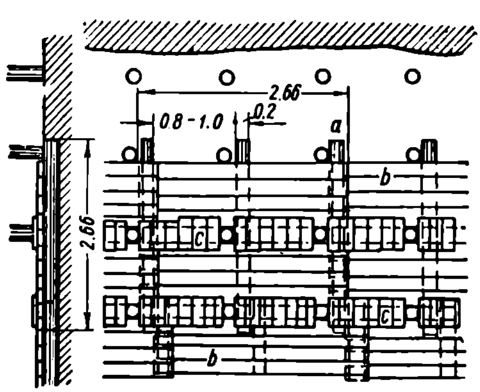



*Fig 267* Temporary support in an inclined slice

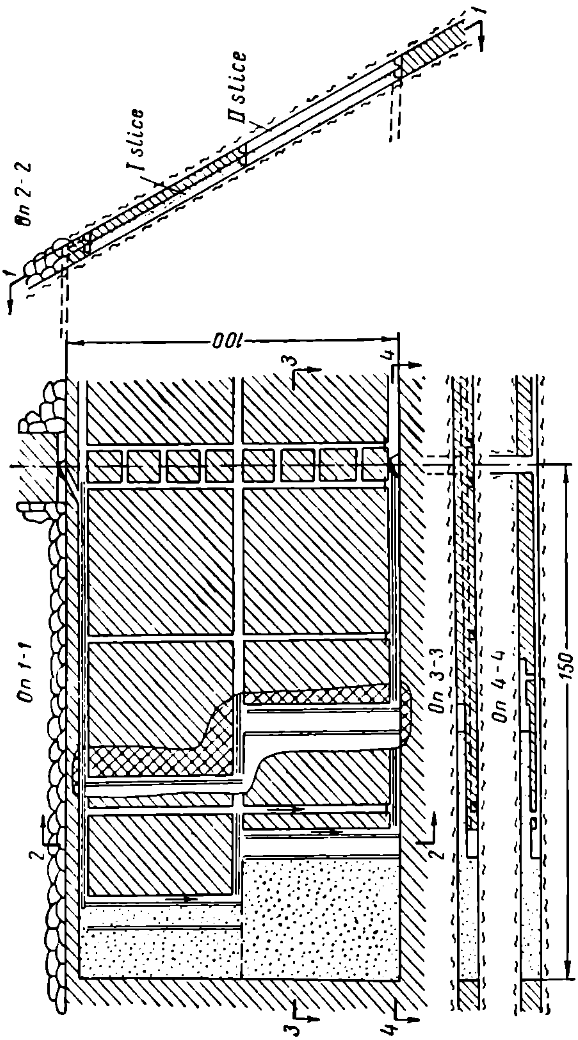


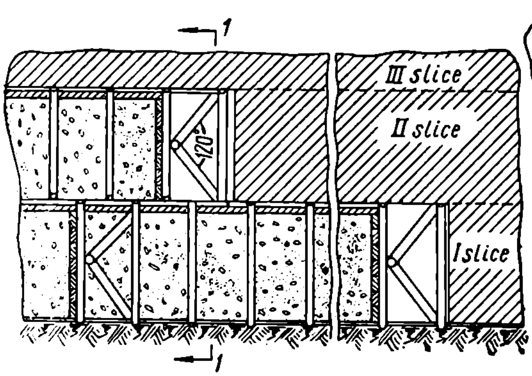

Fig. 268 Inclined ascending slicing-and-filling in a steep seam

## 20. Mining of Steep Seams by Inclined Ascending Slicing-and-Filling

This method is practised in the Kuznetsk coal fields. A 100-metre level is divided into two (Fig. 268) and three sublevels. Depending on its thickness, the bed is separated into two or three slices which are extracted in the ascending order.

Working sections or blocks may be unilateral (Fig. 268) or bilateral, their size on strike equalling 150-200 metres in the first case and 250-300 metres in the second. The disposition of development workings is depicted in Fig. 268.

The upper slices are usually extracted 15-25 metres ahead of the underlying ones.



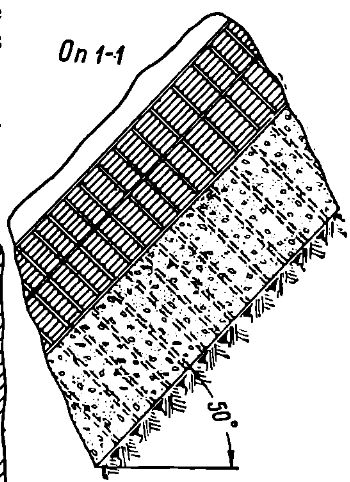



*Fig. 269.* Lagging-off to hold fill at the face

Since mining conditions in individual slices differ, the overlying slices are sometimes designed of lesser thickness.

For example, in a seam divided into three slices the lower one is 2.5-4 metres thick, the central—2.3-3.6 and the upper—2-3.2 metres.

The mined-out space in all of the slices is stowed with fill, and it is only the top that may be worked with caving.

The extraction of an individual slice with filling is effected by the method applying to the case depicted in Fig. 254. The production face of the lower sublevel is 18-24 metres ahead of the upper one. This corresponds to two-three fill "runs" or "intervals".

In the conditions prevailing in steeply dipping beds the filling material in the worked-out area tends to slide down at an angle of repose. Therefore, to hold the edge of the fill mass in place near the face, which in this instance stretches along the dip, a reliable lagging-off is arranged, reinforced by struts, stays and even rafter timbering (Fig. 269). A particularly thorough support is needed near the "banks" of the solid coal mass, its overhanging portions in the upper part of the face (Fig. 270).

Proper application of this method, as shown by Fig. 268, reduces mining losses of coal to 12-15 per cent.

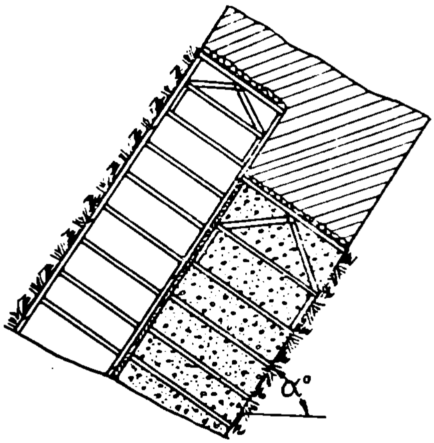


*Fig. 270.* Rafter edge timbering

As far as the geological conditions are concerned, the above-mentioned method has restricted areas of application and this for the following reasons. The thicker the seam the more numerous the slices into which it has to be separated and the greater the discontinuity in the coal block overlying the slice under extraction at the given moment. This is paralleled by growing difficulties in coal extraction and increased losses of the mineral. For this reason it is recommended to employ this system in working seams requiring division into not more than three slices. Ascending slicing is also limited by the angle of dip, for when it exceeds approximately 60°, the stability of coal in the upper slices cannot be secured even by complete filling.

## 21. Inclined Downward Slicing-and-Filling in Steeply Dipping Seams

This method has been used in mining a section of a seam at Mine No. 18b of the Emanzhelinsk district (Chelyabinsk coal fields). The normal thickness of the seam was 20-30 metres, but in the section under discussion it was as high as 35-50 metres because of an anticlinal turn (Fig. 271). A part of the seam, near its outcrop, was extracted by the open-cut method down to the depth of 30 metres. The height of the level interval was 42 metres. Crosscuts were run from main level entry *6* and from these common (mother) entries *5* with slice entries *4* with cross headings over them. The entries and cross headings were connected by rise headings every 5-6 metres.

The inclined slices were 2-2.5 metres thick. Each of them was mined by the retreating method to the crosscut. To start the actual extraction a break-through with the bottom of the open-pit was made near the boundary of the working section. Coal was drawn by "strips"

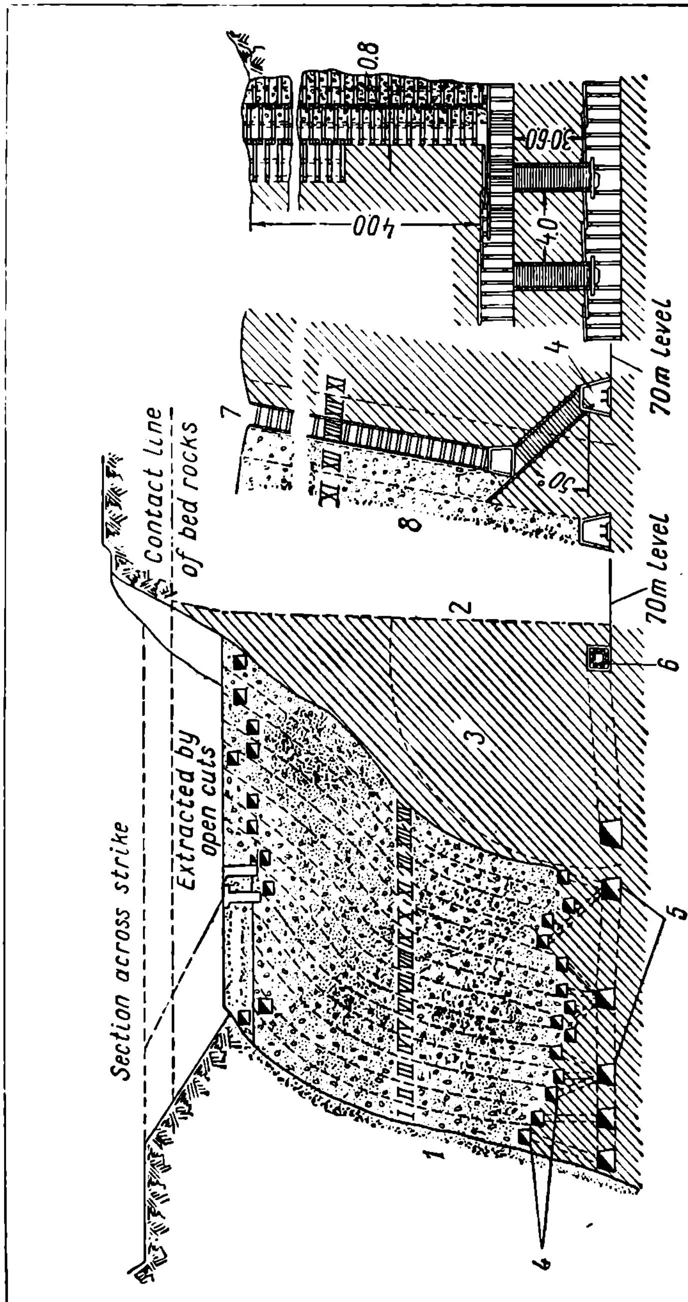



*Fig. 271.* Inclined ascending slicing-and-filling of a high seam

1—hanging wall; 2—foot wall; 3—protective pillar; 4—slice entries; 5—mother entries; 6—main entry; 7—bottom of the open pit; 8—preceding slices

2 metres wide, from top down. Coal was first blasted loose and then broken with pneumatic hammers. One faceman was engaged in each "strip".

In view of the very steep (70-80°) position of the slice, the filling material in its roof and weak coal, it was necessary to provide a very strong support in the production places (Fig. 272).

A protective canopy was arranged over the breast of the face (bench). Parallel with timbering, a lagging-off for the subsequent filling was made along the central row of posts in the strip under extraction. The erection of the lagging-off and the extraction of the coal strip were completed simultaneously. Timber for the support and lagging-off was supplied from the surface with the aid of rope lines.

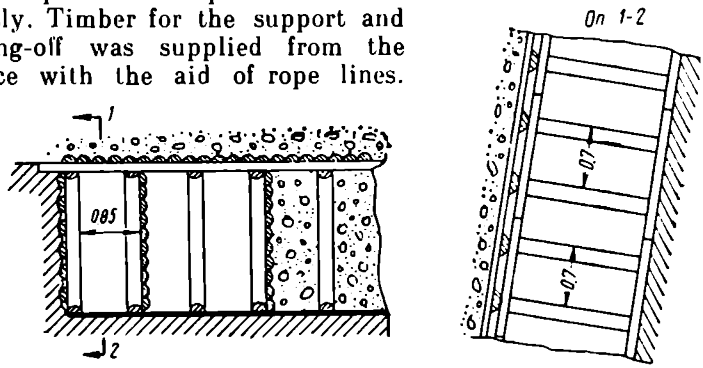



*Fig 272.* Details of a face support

The mine-fill—clay and sandy rocks obtained from the bottom of the pit or the hanging wall of the seam—came from the conveyer by gravity, and for this the mouth of the bore pit over the strip was given a funnel-like shape. The fill of such materials compacted and compressed very well.

The application of this method made it possible to work out the section under discussion in extremely unfavourable conditions—great thickness of the seam, very high dip, weak wall rocks and coal. Although output per faceman in the slice proved high (25-30 tons per shift), the overall efficiency of the section was very low in view of the extremely small footage of active working places in the slice and the fact that only one slice could be mined at a time, and this was the main disadvantage of the method discussed.

## 22. Application Fields of Inclined Slicing

All that has been set forth in Sections 13-21 gives ground to conclude that, in certain conditions, modifications of inclined slicing are suitable for mining thick seams occurring at various angles of dip. But this presupposes more or less regular occurrence and uniform structure of the seam. The workable beds may be separated by gangue partings of considerable thickness.

If the dip is slightly inclined and sloping each slice has to be worked by continuous mechanised (with coal-cutting machines or mine combines and conveyers) faces by the longwall or long pillar on strike methods of mining. In the latter case, the properties of the back in the subjacent slices extracted in the descending order appear to be so favourable that the inclined slicing-and-caving method may be employed even in mining self-igniting coal. That is why this system has been widely used in the Chelyabinsk, Kuznetsk (Leninsk-Kuznetsk and Tom-Usinsk districts), Kizel (Gubakha), Karaganda and other Soviet coal fields in the past few decades.

Generally speaking, the successful mining of thick, slightly inclined seams by the method of *slicing-and-caving* with mechanised walls may be regarded as a major achievement of Soviet coal industry. One past drawback of this system was the abandonment of interslice coal strata.

In mining steep seams, the situation is quite different. Experience and theoretical considerations show that steep beds can be worked by the inclined *slicing-and-filling* method only. Ascending slicing here is admissible only with seams that have to be divided into not more than three slices, that is, of 8-9 metres in thickness, with the dip not exceeding 60°, firm coal that is not subject to quick self-combustion, stable wall rocks and insignificant contraction of the filling. The last slice can be mined with subsequent caving if this is not rendered impossible by the unavoidable dislocation of wall rocks (and the solid mass of filling).

In the case of descending slicing, and provided the properties of the fill are adequate to ensure proper stability of the fill in the roof of the slices, the thickness of workable seams may be quite appreciable (Section 21). But the total working place footage in the slices is so low that one should make a proper choice between inclined and horizontal slicing.

MINING WITH DIAGONAL SLICES

## 23. Diagonal Slicing

The position of slices in respect to the elements characterising the occurrence of a seam is explained in Fig. 261*b*.

In the U.S.S.R. this method was first introduced in the Kuznetsk coal fields in 1936 and underwent tests for a number of years.

Fig. 273 illustrates the mining of a 6-metre-thick seam occurring at 60°. The level, with a vertical interval of 50 metres, is divided into

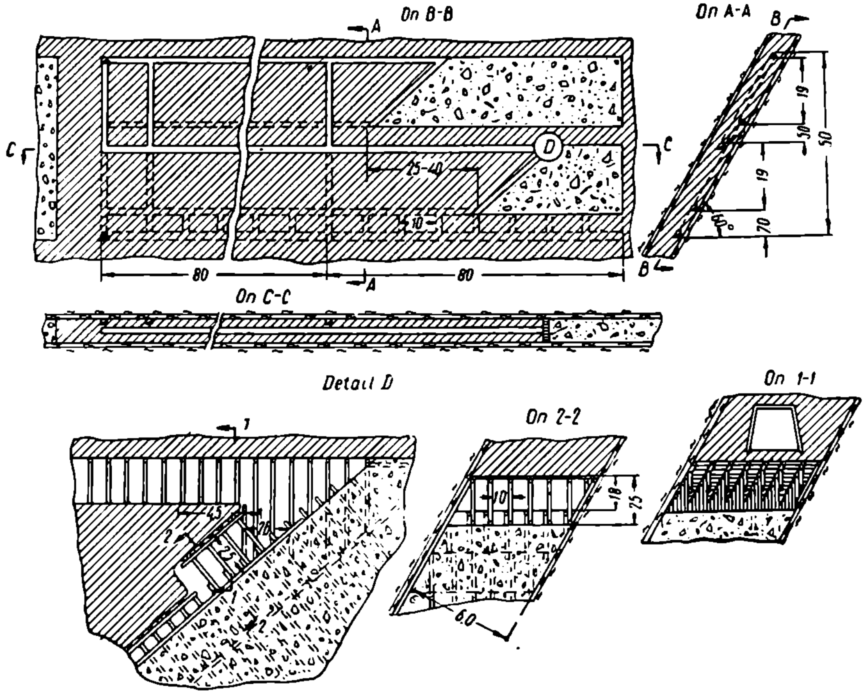



*Fig. 273.* Diagonal slicing

two sublevels. The drawing clearly depicts the progress of development. Stoping in the upper sublevel is done in advance of the faces in the lower sublevel. Diagonal slices lie across the strike, but with an inclination of 35-40° to the horizontal line. This ensures the flow by gravity of broken coal and filling materials in each diagonal slice. To facilitate the breaking of coal and installation of timbering, the slices are usually made about 2-2.5 metres thick.

Production place support in diagonal slices is distinguished by its complexity. The roof of each is made of coal and the bottom of minefill. The timbering includes props erected upright to the plane of the slice and blocked against the headpieces. Since the surface of the fill is not a sufficiently firm base for the props, they are put on sills. If weak, the country rocks of the hanging and foot walls are held in place by inclined struts. To facilitate the setting up and reinforcement of the support, the back of the slice, that is, the overhanging undermined inclined surface of coal, should be as smooth as possible. Since the features of this surface, unlike the ordinary inclined slices, are different from those of the structure of the seam (that is, its bedding and cleavage) the back of each slice has to be levelled out as coal is broken, and that is a labour-consuming operation indeed.

To provide communications between the entries and bring the coal broken in a new slice down, a special "clearance" is left during the mining process. The slice to be filled is not packed completely: free space ("clearance") is left near the roof whose height normally to the plane of the diagonal slice is 0.5-0.7 metre. To this, a board flooring is arranged at a corresponding distance from the back of the slice to limit the height of the fill in the upper section of the slice to be stowed. This same flooring is utilised for dumping coal and serves as a base for the support during the extraction of the next slice. With this method, the top ends of the props of the preceding slice usually stick out from the flooring. Attempts have been made to take advantage of this in order to pull them out with the aid of a lever device and use them again.

Coal is broken with pneumatic hammers after having been blasted loose. Actual breaking is effected by strips 1.5-1.8 metres wide. The broken coal slides down the flooring and iron troughs to the lower entry of a given sublevel, where it is loaded onto a conveyer. The filling material is supplied to the production place from a conveyer set up in the upper entry. Mined-out space cannot be gobbed up above the level of the entry floor with the filling material delivered by gravity alone. Since it is discharged from the conveyer in all directions at the angle of repose, there remain voids near the foot and, especially, near the hanging wall of the seam. The thicker the seam the greater the voids. Therefore, the top portion of each diagonal slice has to be stowed additionally right up to the roof. And this requires special stowing machines (Fig. 274).

Generally speaking, the top portion of a diagonal slice, and especially its connection with the upper entry, is regarded as a critical section. Here it is essential to have a special support consisting of long props and stulls. Sometimes another method is employed instead of the one described above for the support of the interconnection between the diagonal slice and the entry (Fig. 274): sectional or split solid beams made up of squared timbers tightly held together with iron clamps are laid horizontally over the top of the diagonal slice near the entry walls. One end of the beams rests on the floor of the entry and the other on the fill in place. These beams serve as carrier pieces for the support props set up over the diagonal slice.

It cannot be overemphasised that complete and timely supplementary stowing of this area is a paramount prerequisite for the stability of the interconnection between the diagonal slice and the upper entry.

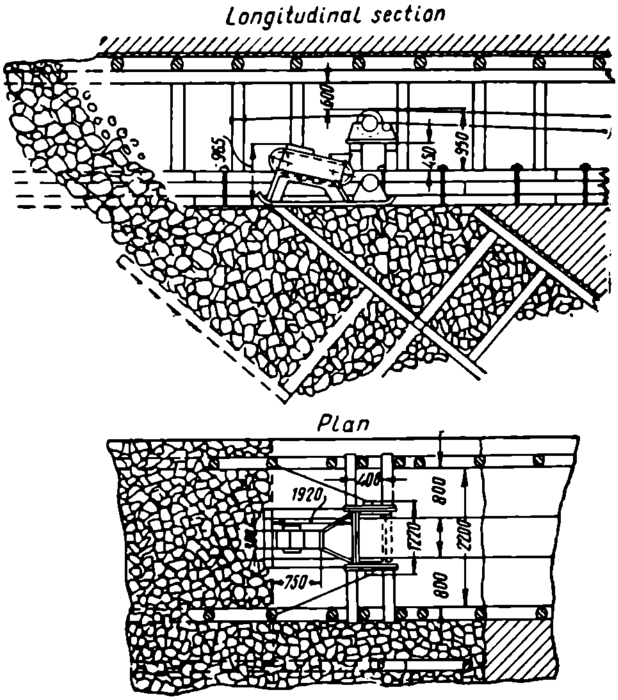



*Fig. 274.* Reinforced timbering of the top portion in a diagonal slice and stowing machine setup

The field tests of diagonal slicing carried out at the mines in the Kuznetsk coal fields in 1936-40 revealed that this system of mining had a number of major drawbacks. Despite complete filling, the total coal losses reached 36-38 per cent, and that is inadmissibly much. Timber consumption was also high: 40-50 cu m per 1,000 tons of coal produced. The constructional elements of the method were complex, while the front of the production faces was small.

Among the advantages of the method are the gravity flow of coal and filling materials in the production faces and the relatively low volume of main and subsidiary development work. A particular stress should be laid on the fact that any modification of this system cannot be successfully applied unless one indispensable condition is met—early and complete filling of the worked-out space. Any, however partial and temporary, lag in filling operations and especially late or incomplete supplementary stowing of the sublevel top portion inevitably cause interruption of work in diagonal slices and possibly a breakdown. Considerable coal losses and the consequent fire hazards, considered the system's drawbacks during field trials in the Kuznetsk coal fields, were in fact due largely to the lag of the filling operations, this being contrary to the very principle underlying the system. When the mines of the Kuznetsk coal fields go over to working thick steep beds with the use of filling, it is not ruled out that the tests of diagonal slicing will be resumed in mining seams dipping at least 55-60° and containing firm coal, with insignificant gangue partings which lie regularly and are geologically unfaulted.

### HORIZONTAL SLICING

## 24. Order of Slicing

The disposition of horizontal slices with respect to individual elements of bed occurrence is shown in Fig. 261*c*. To facilitate work in the production faces, the horizontal slices should be 2-3 metres thick and occasionally somewhat more.

With horizontal slicing, the level is divided mostly into two or several sublevels. The number of slices in each sublevel and, therefore, the number of sublevels in a level are closely related to the accepted order of slicing—ascending (upward) or descending (downward)—as well as to the area in which coal is "undercut" (see below).

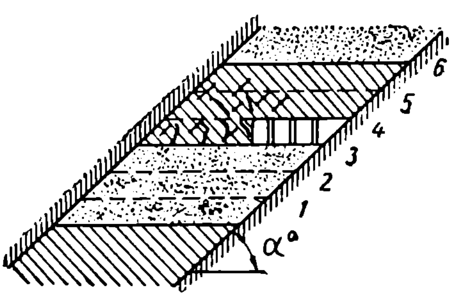


*Fig. 275.* Upward horizontal slicing

Successive upward slicing is possible only with filling. But the fill mass contracts somewhat, this depending on the properties of filling materials and the adopted method of stowing. The larger the number of slices in a sublevel the greater the absolute value of contraction. The result is that after the extraction of lower slices the coal meant for mining by subsequent slices may fracture and begin subsiding (Fig. 275). The fracturing of coal not only creates dangerous working conditions at the face, but may be the cause of fire through spontaneous combustion. These conditions are aggravated still more by the nonuniform settlement of coal, for the mass of fill tends to subside faster near the back, being compressed by the rocks of the hanging wall (Fig. 276). The smaller the angle of dip, the greater the thickness of the bed, the worse the quality of the filling material and the larger the number of slices in a sublevel, the greater is the lack of uniformity. Coal splits intensively in the upper slices, particularly in the last of a given sublevel. A big role in these phenomena is also played by the degree of stability of coal itself.

The effect exercised by the size of the undercut coal area upon the permissible number of slices is seen in Fig. 277, representing in an outline form the vertical sections on strike of two levels (or sublevels) worked by upward horizontal slicing. In both instances pro-

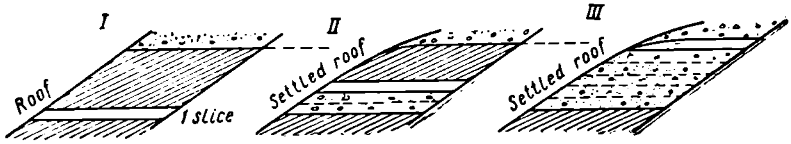



*Fig. 276.* Mine-fill compression under the weight of the subsiding hanging wall rocks

duction places are connected with the lower entry by inclined opening $b$ and with the upper entry by opening $c$. Hence, distance $cb$ characterises the size of the working section or block on strike, but in the first case this distance is small (for instance, 10 metres) and in the second it is considerable—for example, 50-100 metres. In

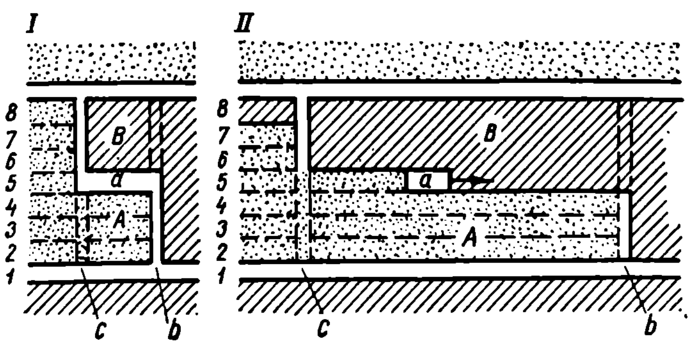



*Fig. 277.* The effect of the size of the working section on strike upon the stability of the undercut solid mass of coal

both cases mined-out and filled space $A$ is overlaid by a solid mass of coal $B$ in the shape of a cantilever (true, this cantilever is in some measure held in place by adhesion with the rocks of the foot and hanging walls and rests upon the resilient ground formed by the mass of fill). But it is clear that, all other conditions being equal, the mass of coal in the shape of a shorter cantilever will be much more stable and consequently pattern $I$, again provided all other conditions are equal, will allow adopting a greater number of slices in the sublevel than does pattern *II*. Thus, the set of factors capable of influencing the number of horizontal slices in a sublevel with ascending slicing is distinguished by its extreme complexity. This explains the big choice of number of slices that can be worked in practical conditions (see below). The greater the number of slices in a sublevel, the smaller is the number of sublevels and the simpler and less costly their development.

When a level is divided into sublevels, the latter are worked from top down. Sometimes, when the sublevel interval is small, it is possible simultaneously to mine two and more sublevels, but the mining phases in the upper ones still have to precede those in the lower sublevels.

In the case of *downward* horizontal slicing, both the caving of the roof and the filling of the mined-out areas are feasible. It is quite common in mining ore bodies to work thick deposits by horizontal slices in a descending order with subsequent caving. But this is practised rarely and only in isolated cases in exploiting coal seams (see Section 31).

Downward horizontal slicing-and-filling has the following disadvantages: 1) the roof of each slice contains a mass of fill which requires particularly strong support; 2) the slices are not undercut, and this complicates the breaking of coal.

On the other hand, this order of slicing also has considerable advantages: 1) it completely eliminates the fracturing and subsidence of coal, as noted above in dealing with the ascending order, and this broadens the margin of safety and is the best guarantee against fires caused by the spontaneous combustion of coal; 2) it does away with the hazards attending the undercutting of a large coal area, and in this respect there are no obstacles towards increasing the size of a working section or block on strike, which makes it possible to widen the front of active production faces. Moreover, preliminary timbering opens up the possibility of working unhindered under the mass of fill.

The result is that in recent years the method of mining thick beds by horizontal downward slicing with preliminary timbering has been constantly gaining ground, with the slices being worked by continuous faces. Metal wire netting can be used (Section 26) as preliminary timbering too.

The cross-sectional shape of a horizontal slice depends on the pitch of a seam. The smaller it is the sharper the corners of the slice at the floor and the roof of the bed, and this complicates the drawing of coal and face timbering. For this reason horizontal slicing is more effective in working of steep beds. But in the case of medium dip the following approach is also possible (Fig. 278): when extracting any of the slices—for example, *II*—coal prism *1* is left in it near the roof of the seam and, conversely, prism *2* is drawn at the bottom of the next slice subsequently packed with waste during the filling of slice *II*. Analogously, in mining slice *III* prism *3* is left and prism *4* extracted, and prisms *1* and *2* are packed, etc., during the filling of this slice. This approach facilitates both the drawing of coal and the building of fill.

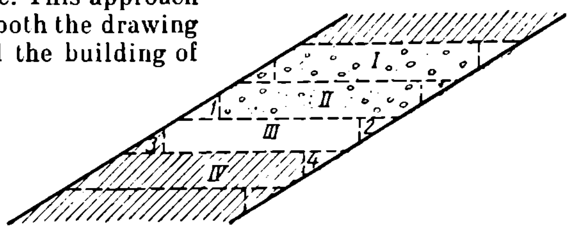



*Fig. 278.* Slice extraction at the bottom and back of the seam

The contours of a horizontal slice in plane may obviously be quite variable in nature. This also makes the system suitable for exploiting extremely irregular deposits.

In the Kuznetsk coal fields horizontal slicing is notably used for mining anticlinal and synclinal turns of folds with inclined axes, where horizontal contours of the slices are of irregular form.

Constructionally, horizontal slicing has a variety of modifications. The most typical are described below.

## 25. Upward or Ascending Horizontal Slicing

When mining steep thick beds with filling was introduced in the Kuznetsk coal fields, there was a method designated locally as "zonal" extraction. This incorrect but brief term denotes the method of upward horizontal slicing-and-filling with the use of so-called square sets.

Fig. 279 illustrates the application of this method in working the Moshchny Seam. The latter is remarkable for its outstanding thickness (usually 14-16 metres) and the purity of coal. The bottom of the bed contains an intercalation of coal-clay shale 0.3-0.5 metre thick. The floor of the bed is weak, the roof strong and the coal hard.

A level with an interval of about 30-50 metres was cut into blocks extending on strike over 50 metres or so. The number of such simultaneously mined blocks was determined by the production programme. The blocks were extracted by working sections (locally—zones), their size on strike being about 10 metres (Fig. 279).

These zones were worked by upward horizontal slicing, with each slice extracted in the direction from the foot to the hanging wall.

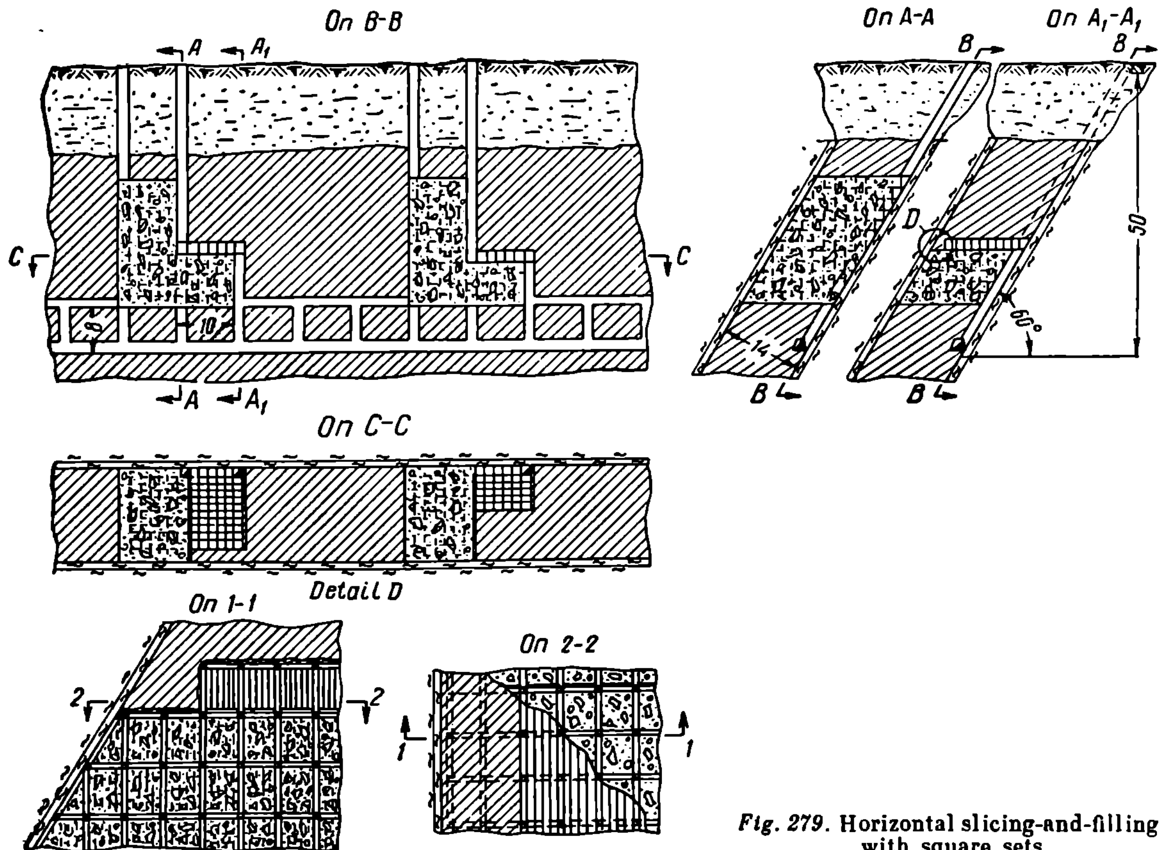



*Fig. 279.* **Horizontal slicing-and-filling with square sets**

The slices were 2.2 metres thick. The mining method involved the use of *square sets* (their description is given in Chapter XXI, Section 5) and complete filling.

In the example under discussion coal was broken by explosives. When the drawing of coal in the two first slices from the bottom of the zone was nearing its completion, work began on filling the first slice. When the first slice was filled, work was started on extracting the third, and the second was filled, etc. Actually, however, filling operations were often allowed to lag and this tended to hamper the regular delivery of planned tonnages.

Coal and filling materials were transported within the boundaries of slices in wheel-barrows and mine cars, but these can very well be replaced by conveyers and stowing machines. Coal was hauled to a dumping chute in the section bordering on the one under extraction. As the drawing of slices progressed, this dumping chute was gradually extended. The filling material was supplied through another chute on the other side of the worked section, which during the extraction of the preceding one served as a dump for coal.

Mine timber was delivered to the slice working places by chutes downwards or upwards, depending on what slices were mined at the moment. The chutes were provided with ladder compartments.

The advantages of the method are: 1) its possible application in a variety of conditions; 2) high degree of safety; 3) convenient conditions for facemen to work in; 4) relatively small area of undercut coal, a factor favourable for the overlying mass of coal in place (see Fig. 271).

The principal disadvantages are: 1) simultaneous use of complete filling and square sets, which is very costly; 2) complexity of support, which requires much labour force and timber (50-55 cu m per 1,000 tons of coal); 3) haulage of coal and filling materials along level workings in each slice; 4) low working face footage, both in the slices and in the block as a whole.

In view of these drawbacks, it is held at present that the "zonal" system has failed to stand the test of field experience, but in the author's opinion, it is not yet entirely ruled out that the question of its applicability in conditions of complete and adequate mechanisation will not come up again.

Figs 280 and 281 are illustrative of the *horizontal slicing with hydraulic or float fill* of the Reden Seam in the Dabrowa coal fields (Poland). Being 10-20 metres thick it dips at 18-20°. Coal is extracted by blocks extending over 400 metres on strike (Fig. 280). The inclined height of the level interval is 200 metres. The horizontal slice is 4 metres thick. The level has about 15 slices and is not divided into sublevels. A slope with a manway is arranged in the centre of the block. The slices are mined in ascending order. A capping of coal

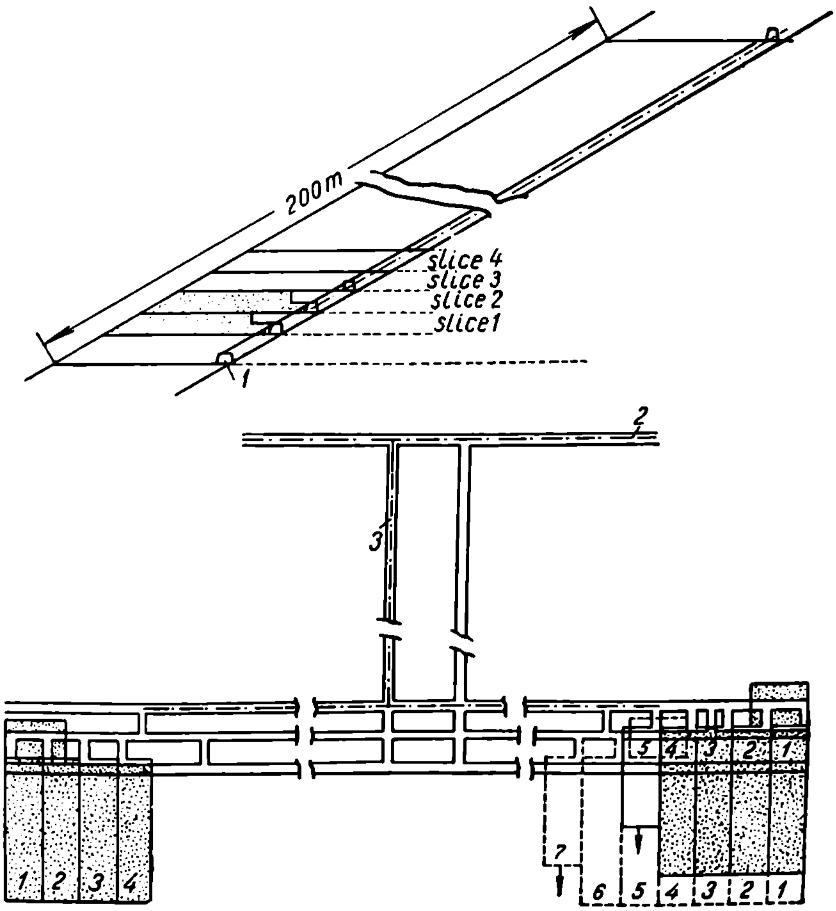



*Fig.* 2 . Horizontal slicing with hydraulic fill
1—main haulageway; 2—airway; 3—mine slope

6 metres thick is provided over the main haulageway. As a rule, the slices are drawn from the boundaries of the block to the slope. The slices are mined by crosscuts. Since the production faces advance across the strike, this method of extracting slices is usually termed "*transverse*".

One of the modifications used in extracting and filling crosscuts is shown in Fig. 281. The width of crosscuts is 8 metres and the height 4.5 metres. During the coal-mining process each crosscut was serviced by two entries: *a* for the haulage of coal produced and *b* for ventilation. At a later date entry *b* is used for laying flushing pipes. The support in crosscuts is illustrated in Fig. 281. From the disposition of timber caps one can see that the crosscut is first worked at a width of 5 metres, towards the hanging and the foot walls, and then the remaining 3 metres are extracted by driving the faces along the strike in the direction of the formerly mined and already filled adjacent crosscut. In approaching the latter, the amount of coal that has to be abandoned is not large, and that in spite of hydraulic filling; there is only a wedge of coal that is left in the end portion of the crosscut. The same drawing shows knee-braces employed to strengthen the roof of the seam. The flushing pipes are laid along entry *b*, situated on the level of the crosscut roof, this making it possible to bring the pipes right to the top of the working. Timber is abandoned in the fill or removed but partially. The crosscut is filled from the back of the seam.

In order to distribute the filling material uniformly over the entire area of the crosscut, the end of the flushing pipe is turned now to the right now to the left. As more fill is delivered to the place, the pipes are shortened. The pipes are handled from special platforms arranged on the timber pieces of the crosscut. In entry *a*, near the crosscut to

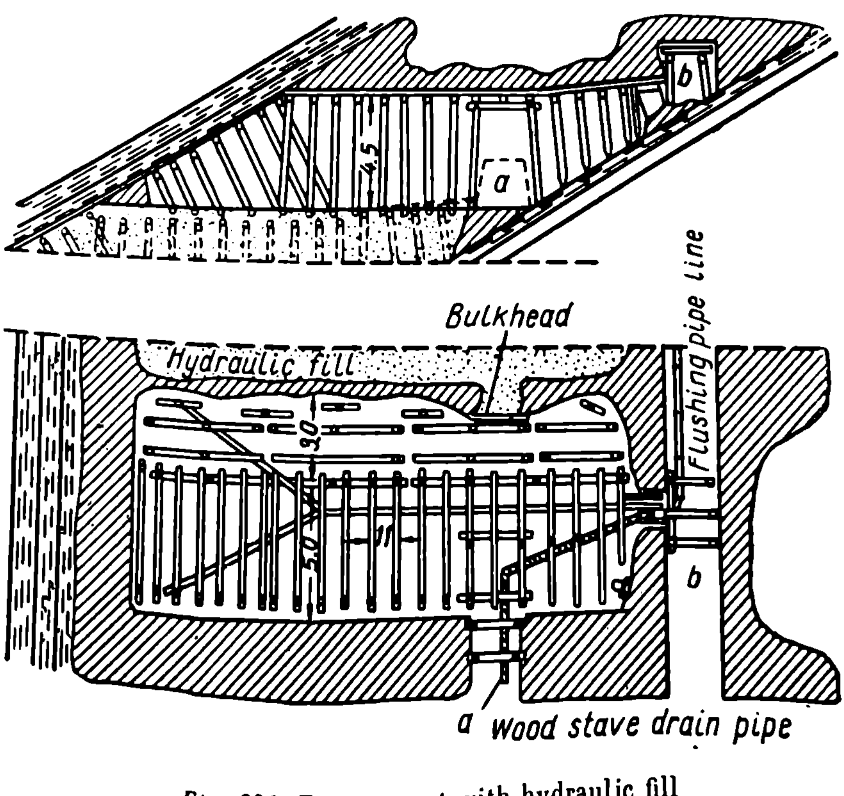



*Fig. 281.* Face support with hydraulic fill

be filled, a bulkhead is built to keep the fill in place. The bulkhead is pierced by a stave pipe with perforations through which the water emitted by the fill runs out.

## 26. Horizontal Descending Slicing

The main points characterising this method are illustrated by Fig. 282, which depicts mining of a thick (10-14 metres) steep seam in the Kuznetsk coal fields.

The blocks on strike are 150-200 metres long. Horizontal slices of about 3 metres in thickness are extracted in downward order. Development openings for communications with working places, dumping of coal, delivery of filling material, ventilation, etc., are made in the central portion of the block which, therefore, is a two-way (bilateral) one. It would be difficult to maintain dumping chutes for the filling material in the already stowed portion of the level and for this reason a coal pillar is left in the centre of the block to protect the above-mentioned development openings, or else lateral workings are driven in the foot wall of the seam (Fig. 282).

The slices are extracted by the retreating method, that is, from the block boundary towards its centre. Slice entries are driven to develop the slices for actual mining. In thick beds slice entries are run at the foot and hanging walls (Fig. 282), while beds of minor thickness are serviced by one slice entry. Each slice is extracted by production faces advancing across the strike. The length of working faces is thus determined by the lateral thickness of the bed.

Mining by longwalls extending on strike with the view to increasing the total footage of working places has the following disadvantages (Fig. 283): 1) the considerable length of time required for developing such a wall; 2) complex nature of timbering and difficulties presented by its mechanisation; 3) short period of actual mining; 4) difficult coordination of coal-breaking and filling operations common to a longwall; 5) difficulties engendered by high rock pressure in a longwall run along the strike.

The slice can be worked in a variety of ways, this depending on the type of mining equipment, timbering and the mode of filling.

Since walls are very short, no coal-cutting machines or mine combines can be employed in them and coal is mined chiefly by blasting and ultimately broken by pneumatic hammers.

Coal in the wall (Fig. 284) is loaded onto short chain-and-flight face conveyer *7* and is then fed to chain conveyer *6* for transportation along the entry to a coal dumping chute.

With blasting employed for mining in a horizontal slice coal can be successfully *self-loaded* onto a face conveyer.

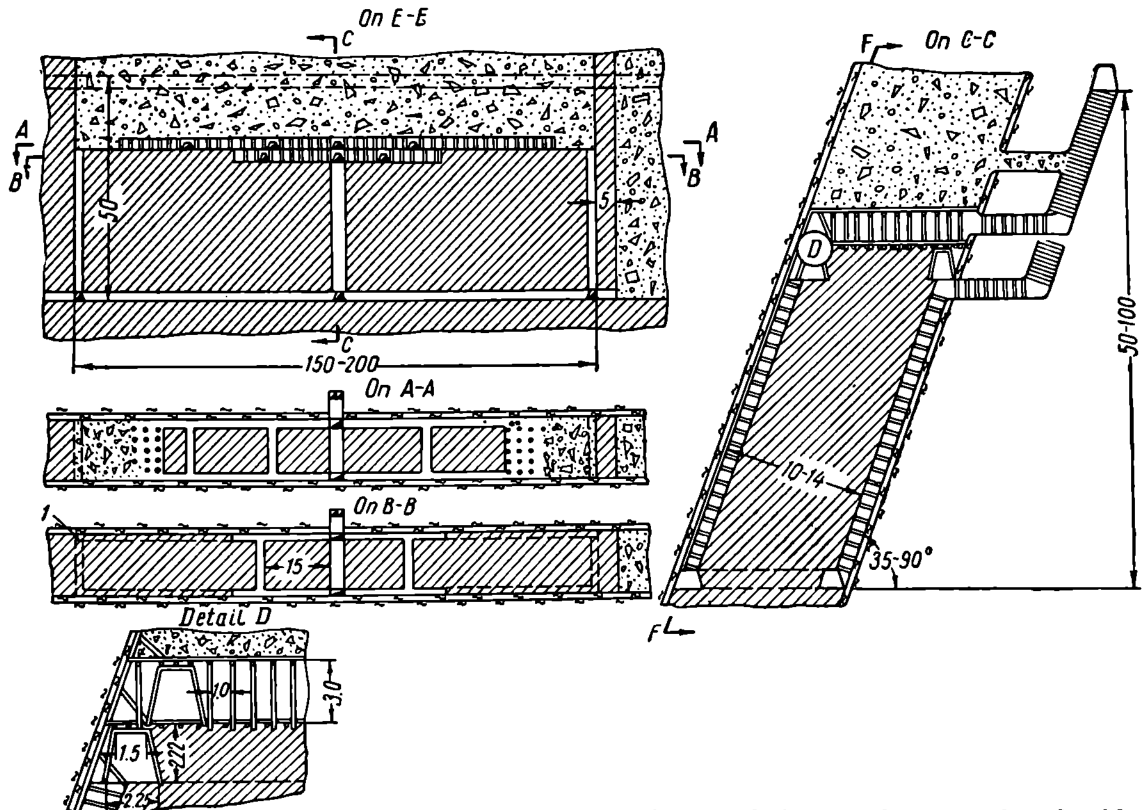



Fig. 282. Horizontal slicing-and-filling in the Kuznetsk coal fields

To accomplish this, prior to the shooting of the charges, deflecting shields of iron sheets or wire netting are suspended from the last row of timber posts near the conveyer and from these blasted coal is thrown back directly onto an active flight conveyer.

If the face of the wall is run not across the strike, as shown in Fig. 284, but obliquely, at an angle of approximately 45° to the strike,

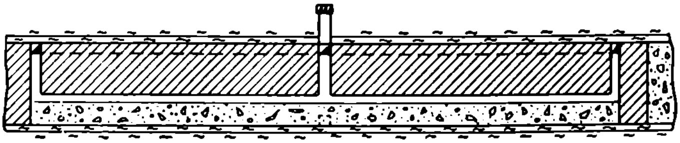


*Fig. 283.* Horizontal slicing with a longwall advancing on strike

the length of the wall and its output per cycle tend to increase. On the other hand, the timbering of the interconnection between the wall and the slice entries is made more difficult.

Downward horizontal slicing-and-filling is distinguished by the fact that there is coal at the bottom of each slice, while the roof is

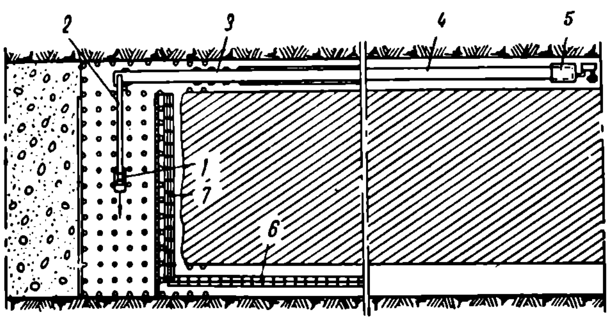



*Fig. 284.* Production face in a horizontal slice with mechanised stowing

made of a mass of fill belonging to the superjacent slice. This necessitates special measures to prevent the fill from falling through into the working slice. That is needed not only to preclude any deterioration in the stability of the fill lying over the working slice, but mainly to avoid any dilution of coal with the filling material.

For this reason timber or wire-net floorings are arranged between individual slices which are sometimes connected with preliminary timbering, as mentioned in Section 17.

Fig. 285 is illustrative of one of the construction types of preliminary timbering. Production faces of slices are supported by rows

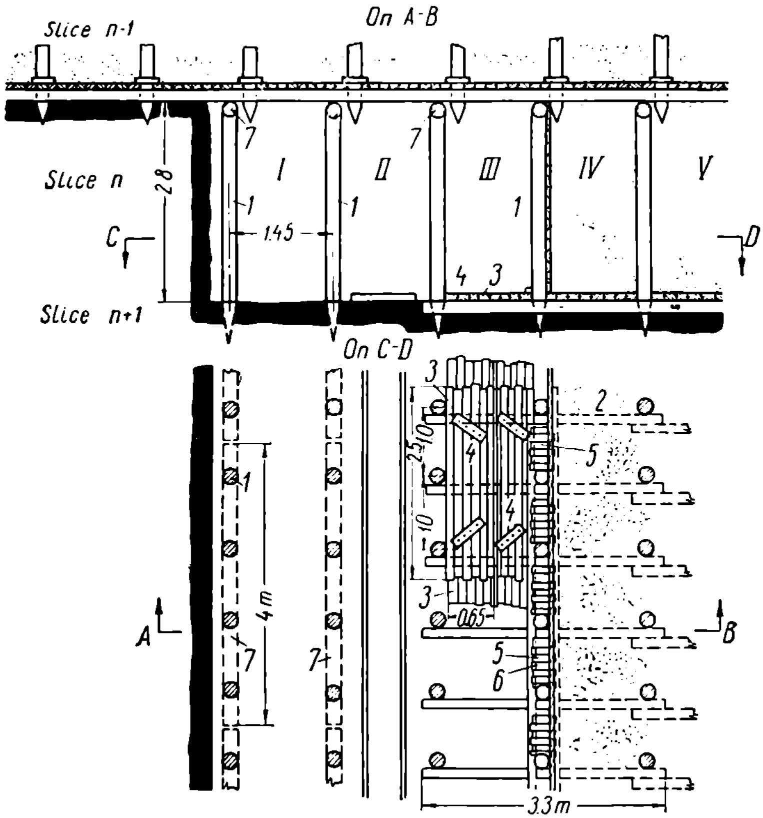



*Fig. 285.* Preliminary timbering

of posts *1* spaced 1.45 metres. The interval between posts in the row is 1 metre. At the phase of the work set forth in Fig. 285, space interval *II* accommodates a conveyer; space interval *III* is being filled; and interval *IV* and those coming after it are already filled.

To install preliminary timbering, ditches are cut out at one-metre intervals in the bottom of working slice *n* to accommodate 3.3-metre-long sills (round or sawn timbers) *2*. The top surface of these sills is flush with the bottom of the slice, making the flooring lie directly on coal. The flooring is made of boards *3* which, to speed up their laying underground, are preliminarily joined on the surface into panels *4*, 2.5 metres long and 0.65 metre wide. Interstices are left between these board panels, along the runs of posts, and they are closed by short plates *5*, placed on boards 6 to prevent the fill from falling through.

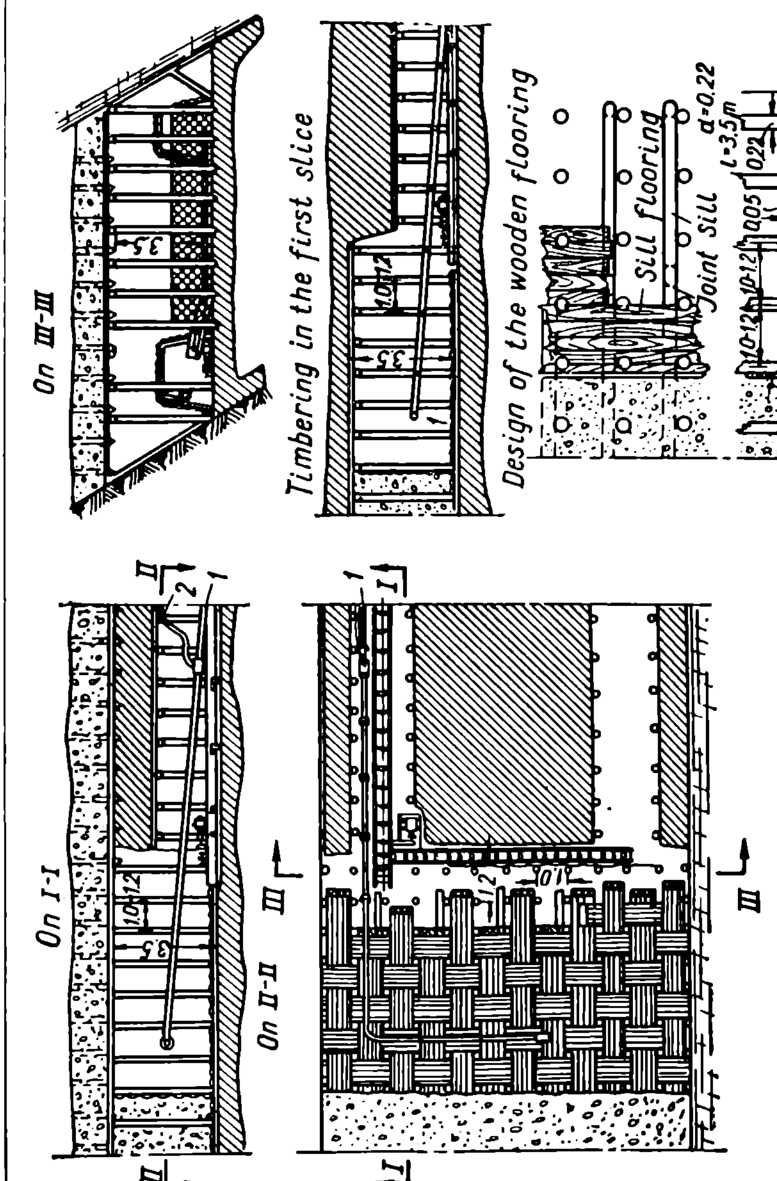



**Fig. 286.** Extraction of coal at the face of a horizontal slice with pneumatic filling

With the progress of coal extraction pick-up caps 7, each 4 metres long, are put up against the roof of the slice, with four posts *1* set up under each. As the fill is liable to contract, the lower ends of the posts are tapered with a view to making them pliable.

Preliminary timbering tends greatly to simplify the support of a working slice, and this enhances the efficiency of facemen. The flooring protects the face from inrushes and the fall of filling material, which is of importance for promoting safety and maintaining the purity of coal.

To prevent spillage of coal, a *wire netting* (Fig. 286) can be employed instead of timber flooring. The diameter of the wire is 2 mm, with meshes $20 \times 20$ mm. To the face the netting is delivered in rolls. When it is laid out, the rolls are spread out between rows of posts—some of them along the strike, others across it, with the "strips" of the netting overlapping each other. The result is a solid flooring. The use of such wire floorings considerably cuts consumption of mine timber.

The fill can be built with the aid of stowing machines or the pipes of a special compressed-air unit.

The operation of a stowing machine in the face of a slice is illustrated in Fig. 284. Feeder *5* supplies the filling material to belt gate conveyer *4* set up in the slice entry, and from there it is loaded onto intermediate sectional conveyer *3*, then goes to face conveyer *2* and is finally fed into throwing machine *1*. The space interval of filling is 6-8 metres.

Fig. 286 shows how filling materials are supplied via pipes from a stationary filling plant. Filling pipeline *1* in the face is in a somewhat uplifted position, this increasing the distance of the throw. A deflector is used to direct the flow of filling material. The space interval of filling on strike is 6-8 metres. When so required, the filling pipeline at the face is shortened. A board lagging-off is provided to separate the rib to be filled from the active stope space. In order to eliminate dust formation and to increase the compactness of the fill, the latter is sprayed with water. Supplied through pipe *2*, water is fed into the filling pipeline *1* via a special sleeve, entrained by the air current and then brought to the breast of the face.

The timbering of the working places is illustrated by the figures. To reduce mine

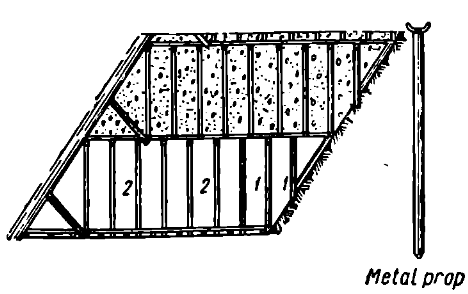



*Fig. 287.* Metal posts face support in a horizontal slice

timber consumption in horizontal slicing, B. Skory has proposed using metal posts, whose design allows their removal from the fill for re-use, for the support of working places. A metal tubular post (Fig. 287) is furnished with two bosses hinged to its top end to support round timber headpieces. The face of the slice is supported partly by timber *1* and partly by metal posts *2*. The latter can be pulled out from the mass of fill in the overlying slice with the aid of a tugger hoist or the driving unit of a coal cutter. The hinged bosses of the post are folded for this purpose and do not hinder the withdrawal of the post.

## 27. Application Fields of Horizontal Slicing

The method is suitable for mining thick beds and deposits of any shape and thickness, containing both hard and weak coal, self-igniting and emitting firedamp.

The stability of wall rocks is of no particular importance. In gently inclined seams horizontal slicing presents some inconvenience. It is much better suited for the exploitation of highly dipping beds and less so for the sloping beds. If a deposit is particularly large, horizontal slicing is possible even when it is flat. In drawing coal from the faces of horizontal slices one comes across diverse coal benches and gangue intercalations (if there are such), which lie obliquely with respect to the coal face. These partings have to be segregated or, if they are too thick, by-passed through additional development and subsidiary openings.

Consequently, except for the above-mentioned reservations, horizontal slicing in its diverse modifications may be used in various conditions of occurrence. In short, it is universal.

But to reduce the amount of labour consumed in its employment it requires, as much as in the case of low seams, thorough mechanisation of basic mining operations. Moreover, since filling is of prime importance in horizontal slicing, equally much attention has to be attached during the preparation of plans and their implementation both to the mechanisation of coal extraction and its transportation and to the whole cycle of filling operations. With the increasing switch-over to mining coal deposits with complete filling there will be a great deal of work to do in improving horizontal slicing practised on the basis of complex mechanisation.

## 28. Fundamentals of Mining by Transversely Inclined Slices

It has already been said (Section 12) that *transversely inclined* slices are the slices whose spatial disposition is outlined in Fig. 261*d*. The simplest way of forming an adequate idea of the pattern characterising this method of mining is to compare it with horizontal slicing in the direction of the strike (that is, when the production face of each slice lies across the strike, see Fig. 282). In mining by transversely inclined slices the general order of development is identical to that in horizontal slicing, but the slices, instead of lying horizontally, are inclined at an angle of about 30° to the horizontal plane, and this inclination is generally in the direction opposite to the dip of the bed. Consequently, as pointed out before (Fig. 288),

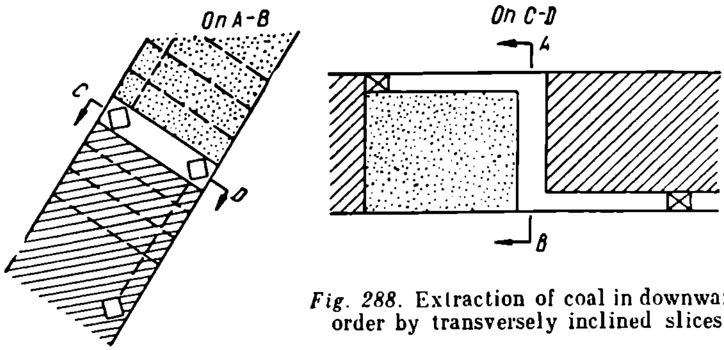



*Fig. 288.* Extraction of coal in downward order by transversely inclined slices

when a seam dips at 60° the transversely inclined slices appear to lie normal to the bedding of rocks. The reasons for this position of slices are to be sought in the desire to ensure the gravity flow of coal and filling materials in the working faces. At the bottom of each slice, near the foot wall, a butt entry is driven for the transportation of coal to the dumping chute, while filling materials are supplied through another, which is left at the time of stowing in the upper part of the fill, near the hanging wall. Inasmuch as slices are extracted by the retreating method from the boundary of the block towards the dumping chute, the slice entry and the flight conveyer put up in it become shorter as the working face moves forward. From an airway horizon of a given level the filling material is passed down via the dumping chutes put up near the boundary of the block, and the length of the slice entry used for the transportation of the fill increases as stoping operations advance. Filling material can be delivered either by compressed air or conveyers.

Fig. 288 roughly depicts the pattern of mining by transversely inclined slices with the descending order of extraction. The upward working of these same slices with hydraulic fill is illustrated in Fig. 289. The position of working places and development openings is shown clearly in the drawing. In mining thick, steep beds by this method, the space interval of filling on strike comprises 8-10 metres. To hold the fill in place, the goaf is preliminarily lagged off by a boarding arranged along the face and nailed to the posts of the breaker

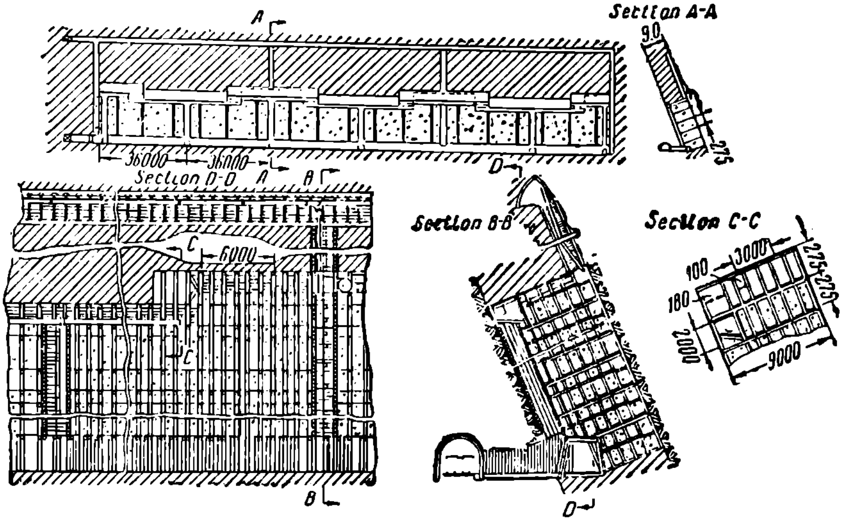



*Fig. 289.* Upward mining by transversely inclined slices

row. A 150-mm stowing pipeline runs along the upper entry. Despite the use of hydraulic fill, consumption of mine timber with this method of mining is as much as 60 cu m per 1,000 tons of coal produced.

Ascending slicing with hydraulic fill is possible only in beds containing strong coal. In the case of weaker coal it is the downward order of slice extraction with filling by flushing or pneumatic fill that is preferred.

Alongside the advantages noted above, such as gravity flow of coal and filling material in the production face and minimal volume of subsidiary development work (blocking-out), the discussed method also presents a number of major drawbacks. Other conditions being equal, the total front of working places is shorter than in horizontal slicing. Setting up of timber in an inclined slice is more difficult than in a horizontal one. Despite the filling, mine timber consumption is exceptionally high. Injuries caused by sliding objects must be reckoned with. All this makes it highly improbable that this method will be widely used.

## 29. Comparative Evaluation of Inclined, Diagonal and Horizontal Slicing

Sections 22, 23 and 27 outline the spheres of application of the methods of mining envisaging the division of a seam into slices of different shapes.

By briefly recapitulating and summarising earlier conclusions, we arrive at the following general inferences.

Slightly inclined thick beds with uniform or relatively uniform structure should be extracted by inclined slices. Since this method admits mining with caving (Section 3), the slices are extracted downward. When a slightly inclined deposit is very thick and its shape and structure are irregular, horizontal slicing may prove to be superior to the inclined, even in conditions of gentle dip.

Both inclined and horizontal slicing is possible in sloping beds. The utilisation of the former method is given preference in the case of persistent structure and occurrence of the seam, availability of considerable gangue intercalations and the not-too-great thickness of the bed.

In the case of high dip, the most universal of the slicing systems is that of horizontal slicing. In beds of moderate thickness, which limits their division to three slices distinguished by persistent attitude and structure, upward inclined slicing-and-filling can be employed. If the fill is readily compressible, the inclined slicing-and-filling method can be applied in mining extremely thick, highly dipping seams, in which the slices are extracted from the hanging to the foot wall, that is, in the descending order.

The possibility of using diagonal slicing is very limited. It is employed in mining steep seams which dip at an angle of not less than 50-55°, are moderately thick with hard coal and devoid of any considerable gangue partings.

In selecting a mining method for a concrete deposit, one should first establish the technical possibilities available for the application of this or that modification and, where several are possible, compare the expected operative, technical and economic results.

## 30. Comparison of Mining Methods with and Without Slicing

When assessing the methods of mining thick beds without their division into slices, that is, by breasting, we see that, as a rule, these methods entail extremely high coal losses, with all the consequences ensuing therefrom. In the final analysis, this is attributed to the fact that the required stability of wall rocks near the working places is achieved by abandoning solid masses of coal in the mined-out area in the shape of rib pillars, stumps, etc., and by leaving coal pillars near various mine workings. Since the active faces are high, the solid masses of coal to be abandoned are of considerable size. In such conditions, mining by breasting in extremely thick beds (over 10 metres) is impossible even technically.

Mining of high faces is both inconvenient and hazardous. To keep a constant watch over the back and the walls of the working space is a difficult thing. Hence the permanent danger of facemen being injured by lumps of coal and rock falling from the breast of the face or its back. Coal should be broken and holes drilled from special scaffoldings or ladders. Another difficult job is to shift and set up long and heavy timber pieces at the faces.

The above cited disadvantages are fully or in a considerable measure eliminated by the slicing methods, and for this reason, they ought to be given preference in exploiting coal deposits of thick beds.

One exception is gently inclined beds of nonuniform thickness (not over 4-5 metres) and irregular attitude and structure, that is, with a varying disposition and thickness of gangue partings that could be worked by pillar mining with stub entries, since in the conditions described above their division into two inclined slices would create considerable difficulties. In the case of high dip, breast mining may be resorted to in extracting beds which, because of their thickness and attitude, can be worked by long pillars on strike or by the shield method of mining.

## 31. Comparison of Methods Employed for Mining Thick Beds with Filling and Caving

Mining of thick coal beds with complete filling has a number of important merits, especially in conditions of high dip.

1. Complete fill appreciably reduces coal losses, thus preserving the country's natural resources.

2. Construction of mines requires certain capital outlays, and the less the waste of the valuable mineral in the mine field the longer their service-life and the smaller the share of amortisation charged against production unit. This also applies to the cost of driving main development openings.

3. One great evil ensuing from large coal losses is the hazard of underground fires caused by the self-ignition of coal. Mining with filling is the principal measure for controlling the danger of such fires. Since some waste of coal is unavoidable even in mining with fill, this method does not entirely eliminate the menace of fires, though it does minimise it. The fires occurring on rare occasions are limited to small areas and can be isolated and controlled more easily than in the case of mining with caving.

4. Mining with complete fill reduces the movement of ground over worked-out areas and rock pressure bearing down on mine workings. It also reduces consumption of timber and the amount of labour expended to set up supports at the face.

After the roof has caved in, the movement of ground over the mined-out area may reach the surface and cause its subsidence, cracks and sinks. All this is attended not only by the vertical settling of surface areas, but also by horizontal shifts. When these accidents assume sizable proportions, it becomes impossible to do any new construction work or to keep edifices and other structures intact. Large cracks and deep sinks tend to destroy the ground surface and sometimes make it impossible to utilise it even for agricultural purposes. Cracks and sinks present a great menace for people and animals and must therefore be either fenced off or backfilled.

5. Cracks and sinks facilitate the infiltration of surface water into underground workings, especially after the thaw or downpours, though the amount of precipitation over the areas affected by cracks and sinks is relatively insignificant. What is dangerous, however, is that water, because of land topography, collects in some other places and then flows in torrents over such areas. Such torrents may bring large volumes of water into underground workings and render it too difficult for the pumping plant to handle and even create a serious danger of flooding.

6. Mining with fill rules out such dangerous phenomena as air bumps. With an open goaf and strong capping subsidences of ground occurring periodically over large areas may be spontaneous and intensive and accompanied by dangerous air blasts which injure men engaged in adjacent mine workings, destroy air doors and partitions and hurl back mine cars and other objects.

The numerous fractures breaching the ground when thick beds are mined with caving of rock walls may intersect underground aquifers whose water then penetrates into underground workings and thus increases the overall influx. This lowers the subsurface water level to a point where the water-bearing strata disappear completely, and that is bad for the water supply.

Mining with fill tends to diminish the significance of the accidents described above and thus helps reduce the inflow of mine water and regulates its distribution throughout the year.

There have been instances of mined-out and unfilled spaces becoming inundated and presenting a danger for headings approaching them from deeper horizons.

7. To prevent and control fires caused by the spontaneous combustion of coal, silting pulp is quite often introduced into the goaf when beds are extracted with caving of wall rocks. It may accumulate in large amounts in caved areas and fractured enclosing rocks and presents a source of dangerous inrushes into the approaching mine workings. Such pulp inrushes have been observed, for example, in the Kuznetsk coal fields during the mining of thick steep beds by rooms. Filling prevents these accidents.

8. Filling is instrumental in improving the ventilation of mine workings. Caving may cause air to leak through the open goaf and fissures in the ground, thus upsetting the adopted ventilation scheme. Such air circulation, difficult to keep track of and control, is capable, as pointed out before, of causing the self-ignition of coal abandoned

in the worked-out areas or rekindling a fire put out before. Circulation of this nature is most intensive in winter months on account of the vast difference in the air temperature underground and on the surface. This circumstance explains why most underground fires caused by the spontaneous combustion of coal occur or, at least, manifest themselves, in winter, especially in regions with cold weather (Kuznetsk and Chelyabinsk coal fields).

Unfilled voids may serve as reservoirs for firedamp which, if there is a sharp drop in barometric pressure or extensive and rapid breaking of the back, may unexpectedly rush into active mine workings.

9. Since in mining with the caving of wall rocks the movement and jointing of ground are quite intensive, this may undermine the superincumbent seams and thus complicate their subsequent extraction. Therefore, such seams should be worked out earlier. The need to extract coal measures consisting of contiguous seams in a definitely set sequence complicates elaboration of time schedules for underground operations. Filling opens up possibilities to take less notice of these factors.

10. Filling reduces the consumption of mine timber.

11. When work with filling is well organised, all routine operations in the mine proceed more systematically and regularly.

The enumerated advantages complete filling has over caving methods can be achieved more or less in accordance with the quality and compactness of the fill, since different grades of filling materials contract differently.

Despite the above-described important merits, the method of complete filling demands setting up a special service whose technical and organisational features and composition are rather complex, require a considerable labour force, and involve capital outlays and running expenses.

To stow completely the worked-out area, the amount of filling material must weigh approximately the same as coal. This means that, along with the extraction and haulage of coal, one has to excavate and transport similar quantities of waste. Since it is impossible to obtain adequate amounts of waste material from underground sources in mining thick beds, it is necessary to open up quarries on the ground surface to excavate the needed materials. In the case of large-scale coal production involving complete filling, the basic operations of excavation, transportation and preparation (crushing, screening, etc.), the delivery to the mine of filling materials, their subsequent underground haulage and stowing in working places have to be thoroughly mechanised.

This is an indispensable prerequisite for enhancing the efficiency of the men engaged in all phases of filling operations and for making these operations economical. Their mechanisation, however, requires considerable capital outlays and running expenses and a special staff of workers, and complicates the problem of management.

There is no doubt that these outlays and working expenses are fully or partially compensated for by the advantages inherent in the complete fill method which, in the final analysis and depending on local conditions, may not only prove technically inevitable or convenient but economically profitable.

Deciding whether any given seam should be worked with filling or caving is a rather complicated matter. In doing it, one should consider the existing local conditions, taken as a whole. Filling operations are closely connected not only with the actual method of mining but with all of the mining operations performed at the mine. That is why deciding whether work is to be done with fill or caving necessitates first elucidating which mining methods should be employed to exploit a given deposit with filling and which with caving and then drawing conclusions on the effect exercised by each method upon the functioning of the mine as a whole from the technical, organisational and economic viewpoints.

In selecting a method for mining a coal deposit, with filling or caving, one has to take into account that the angle of dip of the bed is a prime factor. In the case of slightly inclined or sloping dip, a thick seam can be worked by inclined slicing, each slice being mined by continuous mechanised walls with caving. We have seen earlier that in mining with continuous faces the advantages include reduced coal losses and rapid and complete settling of the roof rocks. This largely counterbalances the numerous shortcomings of the method of mining without fill, referred to above, and makes inclined slicing-and-caving of relatively uniformly occurring thick beds with persistent structure quite possible and feasible. This was the way taken in the U.S.S.R. in developing systems of mining thick, *gently inclined* beds, that is, by inclined descending *slicing-and-caving*.

In mining thick, *highly pitching* beds the situation is quite the reverse, and here the basic feature is the application of *filling*. All the experience so far accumulated definitely prompts to discard the practice of working thick steep seams with the caving of wall rocks.

It should be emphasised that the decision to work with complete filling should be carried out systematically, in full and according to schedule.

Any lag in filling operations leads to extremely objectionable consequences: coal losses, fires, spontaneous caving-in of coal and rocks in production faces, increased consumption of mine timber, reduced coal output, etc. When, in the case of the filling method, stowing operations are behind schedule, this may prove to be more hazardous than the system involving the caving of rocks in the same conditions.

CHAPTER XVI

# UNDERGROUND GASIFICATION OF COAL

## 1. Historical Background and Importance of Underground Gasification of Coal

To be utilised coal is extracted from the bowels of the earth.

But it can be utilised in another, entirely different way—the one involving its conversion into *gas* at the place of its occurrence underground and subsequent use of this gas on the ground surface as fuel or raw material for important chemical products.

The idea of such *underground* gasification of coal was first put forward back in 1888 by the great Russian chemist, D. Mendeleyev, who, having studied underground fires caused by the self-ignition of coal, wrote: "As far as these underground fires in coal beds are concerned, it seems to me that they may be made use of if controlled and managed so that combustion proceed in the same manner as in a gas producer or generator, that is, with low access of air. This would produce carbon monoxide and the bed would give generator or producer gas. Several holes should be bored in a bed, some of them destined for introducing and even blasting air into the bed and others for the exit and even exhaustion (for instance, with the aid of an injector) of combustible gases which can then be readily fed to furnaces even at long distances."

The same idea came to the English scientist, Sir William Ramsay, and was highly appreciated by V. I. Lenin.

In an article entitled "One of the Great Victories of Technology" Lenin wrote that the new method of direct production of gas from coal seams "transforms coal mines into something like huge stills for the production of generator gas. The latter drives gas motors, which make it possible to use twice as much energy contained in coal than it was possible in ordinary steam machines. In turn, gas motors serve to convert mechanical energy into electricity which modern techniques are now capable of transmitting over huge distances.

"The cost of electric power would, as the result of such a technical revolution, decrease to *one-fifth* and perhaps even *one-tenth* of its present level. An immense amount of human labour now employed for extracting and transporting coal could be saved. Use could be made of the poorest and at present unexploited coal deposits. Expenses for lighting and heating of dwelling houses would be greatly reduced.

"The industrial revolution brought about by this discovery would be of tremendous importance.

"But the consequences of this revolution for public social life under the present capitalist system would be quite different from those entailed by this discovery under socialism.

"Under capitalism the 'release' of millions of miners engaged in coal production would inevitably engender mass unemployment, a great deal more misery and deterioration of workers' living standards. The profits from the great invention would go into the pockets of Morgans, Rockefellers, Ryabushinskys, Morozovs and into those of their retinue of lawyers, directors, professors and other lackeys of capitalism."

Continuing, Lenin indicated that under socialism the use of the new method "'releasing' millions of miners would make it possible immediately to shorten the working day *for all* workers from 8 to, say, 7 hours and even less. The 'electrification' of all factories, plants and railroads would make working conditions more hygienic, free millions of workers from smoke, dust and filth, facilitate the conversion of dirty, abominable shops into clean, well-lighted laboratories, worthy of man. The electric lighting and heating of each dwelling house would spare millions of 'household slaves' the need to spend three-quarters of their life in a foul-smelling kitchen."

The first to advance a pattern of mine workings for underground gasification of coal was Prof. B. Bokiy who published it in an article "Ways for Further Progress of Coal Industry in the Donets Basin" in 1921 (*Iron and Coal*, 1925, No. 1).

In 1933-34 experiments on underground gasification of coal were carried out near the towns of Lisichansk and Shakhty in the Donets coal fields, at the Krutovskaya Mine in the Moscow basin, and at the Lenin Mine in the Kuznetsk coal fields.

In February 1935 an experimental mine was opened at Gorlovka (Donets coal fields) and in 1940 a large pilot station for underground gasification of coal was commissioned in the Moscow coal fields.

## 2. Basic Notions on the Methods of Underground Gasification of Coal

There are two basic methods of underground gasification of coal today: 1) through mine openings and 2) from the surface.

The first method (Fig. 290) envisages running two inclined passageways (or large-diameter boreholes) *a*, which are connected by entry *b*. The solid mass of coal delimited by these openings is called *gasification panel.*

Coal is first ignited directly in *kindling* working *b*, and this creates a *fire face*. The process of coal gasification is conducted in such a manner that one of the passageways (airway) lets in the air blast which then passes on to the fire face and, sweeping the bed, burns the coal. Gas escapes to the surface via the second, gas-tapping passageway. The top portions of the passageways are sometimes replaced by boreholes *c*. Consequently, there is no need first to break or crush the bed, for coal is gasified "in situ". Advancing up the rise, the fire face gradually assumes a curved, concave shape (as shown by the dash line in Fig. 290).

The amount of coal in the panel under gasification depends on the size of the panel and the thickness of the bed. In selecting the length of the panel, account is taken of the blasting equipment available (blower or compressor capacity and air delivery head), the chemical composition of the coal and the strength of the roof rocks. Usually the length of the panel ranges from 100 to 300 metres. Long panels are split into sections which are serviced by separate passageways.

During the exploitation of a panel the bottom ends of the passageways are in direct contact with the fire area or space. The air and gas passages are lined with iron pipes, whose ends gradually burn out. To prevent coal near the gas passage from igniting prematurely, the pipes are protected by a waste pack.

It was Soviet specialists who elaborated and introduced another method of underground gasification under which the underground gas producer is prepared and exploited from the ground surface. With this method, the mine openings necessary for the operation of

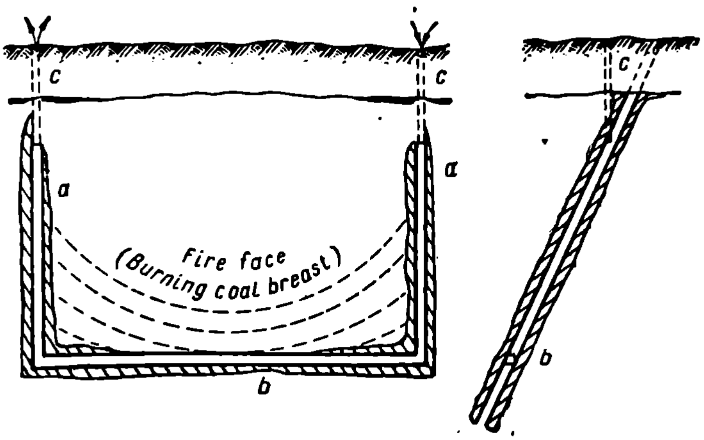



*Fig. 290.* Underground gasification of coal through mine openings

the underground gas producer are replaced by holes bored down from the surface. Some of them serve as openings for the delivery of the air blast and others for the withdrawal of the gas obtained.

To form an initial channel for the gasification process in an underground gas producer prepared by this method, it is necessary to connect the inlet blast and outlet gas holes through the coal bed.

The formation of such a channel or, as this process is usually called, the connection of boreholes through the coal bed, may be effected in different ways. The most widely used is so-called fire infiltration break-through.

If an air blast is forced under a head into a coal bed via one of the two holes bored from the surface, a certain portion of this blast will escape through the second hole. The blasted air will then penetrate through the pores and cracks in the coal bed or, as it is said, infiltrate through the latter.

The velocity of such gas movement depends on the gas permeability of a coal bed. As is known, different coal beds possess a varying degree of gas permeability.

Experience has borne out the feasibility of such an infiltration fire break-through and at present the Moscow suburban station of underground gasification operates exclusively by this method without employing any men underground.

The composition of the gas obtained in an underground gas producer depends on the quantity and quality of the air blast and also on the direction of the air blast and gas currents in the underground generator. With an ordinary air blast, the caloric value of the gas received reaches 1,100 cal/$m^3$. A higher caloric value may be achieved by increasing the proportion of oxygen in the blast, adding steam to it, changing the rates of blast and appropriately orienting the direction of the blast and gas in the underground gas producer.

In general, an underground coal gasification station has the following basic installations: 1) mine openings or boreholes; 2) an air blast unit (compressor blower or an oxygen plant); 3) boiler plant; 4) surface pipelines (steam, air, oxygen and gas); 5) water plant and facilities.

The degree to which this or that section of the station is developed depends on the method and system of gasification, the output of the station and the local consumers. If gas consumers are far from the station, the latter is equipped with an additional gas blowing plant. The Lisichansk station, for example, has special gas blowers to supply remote consumers.

The theory of underground gasification of coal is being further elaborated. Although the types of stations have not yet been fully decided upon, the problem, as a whole, is nearing solution.

CHAPTER XVII

# HYDRAULIC MINING OF COAL

## 1. Basic Notions of Underground Hydraulicking

The considerable successes achieved in hydraulicking by open-cut methods and in hydraulic earth work during the construction of hydropower plants and structures have prompted using this method for underground mining of coal and ores.

On the initiative of Engineer V. Muchnik the first tests in the coal industry were conducted in 1935, in the Kizel coal fields in the Urals, then a pilot *hydraulic mine* was commissioned in the Donets basin. Its operation was suspended by the nazi invasion. After the war experimental hydraulicking was resumed in the Kuznetsk coal fields.

Essentially, the method consists in that coal or any other mineral is loosened in the working places by powerful jets of water from the nozzles of *hydraulic giants* (monitors) under a head of 35-50 atm. The broken and ground coal becomes mixed with water and is transported from the production face to the shaft by the water current along special troughs (sluice boxes) or pipelines laid in underground openings. If the mixture of coal and water (pulp) is brought to the surface by "coal suckers", the winning of coal at the working place, its transportation in underground workings and delivery to the surface are effected by a water stream in a single continuous process, this greatly simplifying the whole pattern of underground and surface mine plants and arrangements and ensuring a high degree of labour efficiency. On the surface coal is segregated from water in special settling tanks or goes to dressing and concentration plants. Having made a full cycle, water is fed back to the mine for further hydraulicking.

## 2. An Example of Underground Coal Hydraulicking

Large-scale experimental hydraulicking of coal has been carried on since 1949 in the Kuznetsk coal fields, at the Tyrganskie Uklony Mine where a steeply pitching seam is being mined (Fig. 291). Since the workable bed is high and contains self-igniting coal, its develop-

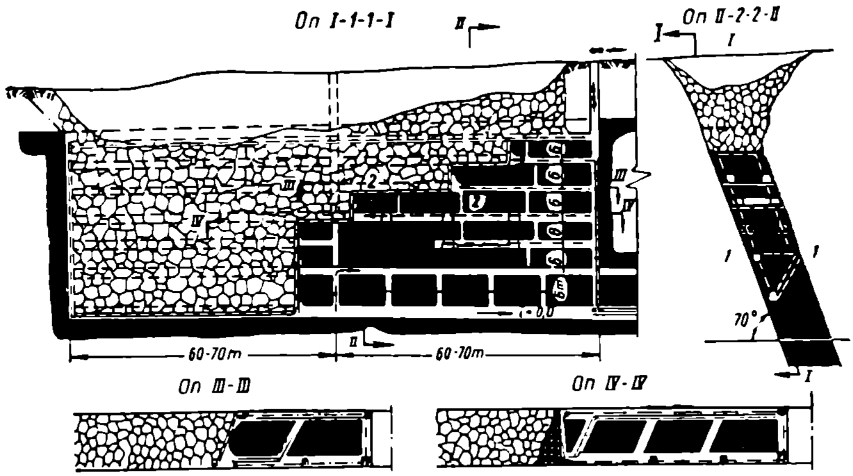



*Fig. 291.* Underground hydraulicking of a thick, highly dipping coal seam

ment for extraction is done via panel (intermediary) crosscuts driven from an entry running along a neighbouring low seam. Fig. 291 is illustrative of the method of mining by subentries, a modification of the method of long pillars on strike employed in this instance.

To facilitate loosening of coal by the hydraulic giant the sublevel intervals are very small—6 metres. Since the mined-out space remains unsupported, *a flexible flooring made of wire netting* is laid to prevent coal loosened by the water jet from mixing with wall rocks. The netting is made of 2-mm steel wire with meshes measuring 20× ×20 mm. It is delivered to the mine in rolls one metre wide and 15-20 metres long. When the flooring, comprised of four wire-net layers is arranged, the ends of the rolls are made to overlap each other.

The flooring covers the top and the sides of the solid coal to be hydraulicked. The flexible flooring is initially laid out as follows. A *slot*, representing a vertical opening across the strike of the bed and over its entire thickness, raised to the height of three sublevels, is cut in the coal seam on the boundary of the working panel. The slot, supported by timber sets, is fitted with a wire-net flooring, and is about one metre wide. In addition to the slot the flexible flooring is also laid on solid coal. For this purpose a horizontal slice of coal is drawn at the level of the upper entry and four layers of wire-netting are laid crosswise upon its bottom. This horizontal slice is also supported by timber sets. As the wire flooring is laid, coal loosened by the water jet in the roof of the slice falls on the ready sections.

Strictly speaking, the excavation of the slot and the drawing of the horizontal slice are operations preceding the actual stoping, since these openings serve only to accommodate the flexible flooring. The sublevels are mined in a descending order and the process begins when the flexible flooring on coal extends over 30 metres. The advance rate of production faces in sublevels should not be less than 15 metres.

Coal is extracted by the hydraulic giant through cuts or stub entries three metres long on strike, from bottom upwards and from the foot to the hanging wall of the bed. After the extraction of each cut, the hydraulic giant is shifted three metres back to start drawing the next cut, while the trough sections for the transportation of coal and the water-supply pipelines are correspondingly shortened.

The headings of the cuts are unsupported. Prior to their extraction the timber of the entry is removed. The part of the flexible flooring hanging down from the back to the floor of the face and thus forming a protective partition separates broken coal from barren rocks falling into the mined-out area. As the extraction of cuts progresses, the flexible flooring thus keeps descending from the roof of the sublevel to its bottom. Good care should be taken to prevent the flooring from shifting onto the bottom of the bed, this being eliminated by the changed order of coal extraction. Since the flexible flooring gradually wears out and falls into disrepair, the wire netting lasts for the extraction of only three sublevels, after which it becomes necessary again to draw a horizontal slice at the level of the third subentry and arrange a new flexible flooring.

Attempts at extraction without the use of wire flooring have resulted in the reduction of labour force but in higher losses of coal.

Coal loosened in production and development faces by the hydraulic giant goes together with water to troughs or sluice boxes, along which it is conveyed to a coal-dumping chute and passed down to the lower entry, where a stationary screen with 60-mm mesh is set up under the dumping chute. Screened or undersize coal is then immediately transported together with water along a line of troughs put up in the lower entry, while oversize coal is first broken in a nearby crusher. To facilitate the transportation of coal by a water stream, the entries are made to slope 0.03-0.05. To the surface the coal is lifted by hydraulic elevators, the high head "coal suckers".

When the coal bed is more than 10 metres thick, two entries are made in each sublevel—one at the foot wall and the other at the hanging wall. If the bed is thinner, it suffices to drive but one entry, in the centre of the seam.

Underground hydraulicking of coal in the Kuznetsk coal fields is practised in a slightly inclined seam at the Severnaya-Polysaevskaya Mine.

The technical and operative indices of underground hydraulicking of coal have yet to be precisely determined, but in the above-named mines labour productivity is two to three times higher than in ordinary mines.

### 3. General Flowsheet for Underground Hydraulicking

In modern and planned hydraulically operated mines all the coal produced is brought to the surface by a hydraulic lift. This is done by specially manufactured high-pressure "coal suckers" operating under a head of 120-130 metres; new ones of a still greater head are now being designed. The flowsheet of a hydraulically operated mine will then be as follows (Fig. 292). Coal entrained by water is delivered along troughs *5* and pipes *6* to pulp collector *9*, set up in the coal-lifting chamber near the shaft. It is then sucked into pulpline *7* and brought by "coal suckers" *8* via a pipeline to the surface and fed to coal-concentrating mill *1*, where it is dressed, dewatered and dried. The cleared water is fed back to pump station *2* on the surface which is equipped with high-pressure pumps, and from these, through hydraulic giants *4*, it is supplied to production face *3*.

The underground hydraulicking of coal is a new progressive method capable of considerably heightening labour efficiency and reducing mining costs.

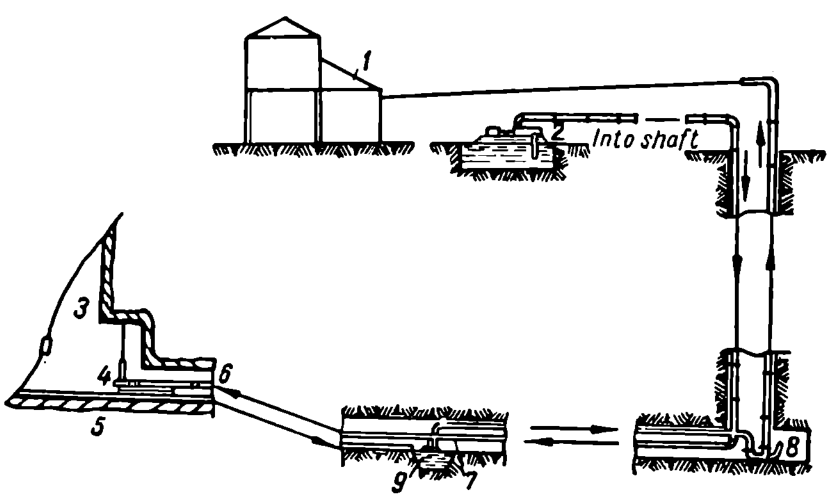



*Fig. 292.* Hydraulically operated mine flowsheet (with hydraulic lift)

*1*—dewatering plant; *2*—high-pressure delivery pump station; *3*—production face; *4*—hydraulic monitor; *5*—coal transporting troughs; *6*—water supply pipeline; *7*—pulpline; *8*—pulp pump; *9*—pulp collector

Technically, the basic merit of underground hydraulicking is the combination of mining and transportation operations achieved with the aid of relatively simple mechanical equipment. The main drawbacks are high power consumption (about 25-30 kwh per ton of coal output, as against the usual 10-12 kwh), increased humidity in mine workings and, with the methods actually employed, considerable losses of coal.

# *DEPOSITS OF NATURAL SALTS*

CHAPTER XVIII

## METHODS OF MINING ROCK AND POTASH SALTS

### 1. Shapes of Rock and Potash Salt Deposits

Natural salt deposits are of sedimentary origin, and for this reason in geologically undisturbed or slightly dislocated regions they occur in beds of diverse thickness dipping at a low angle. One example is the Artyomovsk district of the Donets coal fields where *rock-salt* beds of up to 40 metres in thickness extend regularly and almost horizontally over very large areas. The famous Solikamsk deposit of potash and magnesia salts in the North Urals is also generally flat and spreads over a huge area, although in places it has folds, complicated by displacements with rock ruptures. Valuable minerals here are represented by *sylvinite* (a mixture of potassium chloride KCl, sylvite, with rock salt) and carnallite ($KCl \cdot MgCl_2 \cdot 6H_2O$).

Carnallite also serves as a source of magnesium and its compounds.

As compared to many other rocks, rock and potash salts are distinguished for their high degree of *plasticity*, and for this reason many deposits occur in the shape of *salt domes* or *plugs* (Fig. 293). These original forms of salt occurrence have in all probability appeared as the result of their extrusion, facilitated by the plasticity of salts under the pressure of surrounding rocks. Usually, though not always, salt domes are of an oval shape in plan, their long axis extending over 1.5-3 km, steeply dipping down into the earth crust. During the formation of domes the circumjacent rocks, one may assume, were depressed and somewhat uplifted. In the U.S.S.R. a typical salt dome is the one at Sol-Iletsk (Southern Urals). Its geological structure conforms to the pattern shown in Fig. 293. The extent of the dome along its long axis is about 2 km, and around 1 km along the short. Continuous occurrence of rock salt in this dome has been confirmed directly by exploratory borings made down to a depth of 500 metres from the surface. Judging from data obtained through geophysical

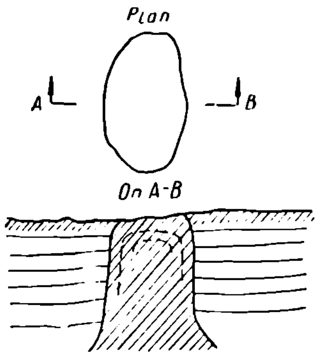



*Fig. 293.* Outline of a salt dome

prospecting, salt occurs down to the depth of not less than 1.5 km.

Many salt domes are of a much more complex geological structure than the one shown in Fig. 293.

A large number of salt plugs, so far not mined, have been found in the course of geological surveys on an extensive territory north of the Caspian sea coast.

The selection of a proper method of working rock salts is influenced mainly by two factors: 1) because of their ready solubility rock salts must be mined so that no water penetrates into mine workings from the ground surface and underground aquifers; 2) the reserves of natural salts are so great that there is practically no need to be too cautious about the losses during their exploitation. As the result of the combination of these two factors, rock and potash salts are almost invariably mined by the method involving *abandonment of support pillars.* The latter help completely eliminate any movement of the ground overlying the mined-out areas. Salt deposits can be worked with continuous faces only in gently inclined low and medium-thick beds, particularly in those of complex structure, for the gangue from partings can then be utilised as a fill for stowing worked-out space, at least partially, through ensuring gradual settlement of roof rocks. In such exceptional conditions of occurrence, salt deposits are mined by methods similar to the methods of working coal seams described earlier.

## 2. Methods Employed for Mining Rock Salt

Rock-salt beds in the Artyomovsk district of the Donets coal fields lie 120-200 metres below the ground surface. The principal of the working beds, the so-called Bryantsevsky bed, is 30-40 metres thick. In the mined areas the angle of dip is about 4°. Salt reserves are practically unlimited.

Either rooms *A* lie parallel to each other in one direction (Fig. 294), or two sets of parallel rooms traverse each other (Fig. 295). Interchamber *support pillars* of salt *a* play the role of columns holding up the superjacent ground and are therefore never recovered. Thick deposits of rock salt are mined so as to prevent the roof from caving in. Such cavings are not only very dangerous by themselves, especially if one bears in mind the immense width and height of the workings,

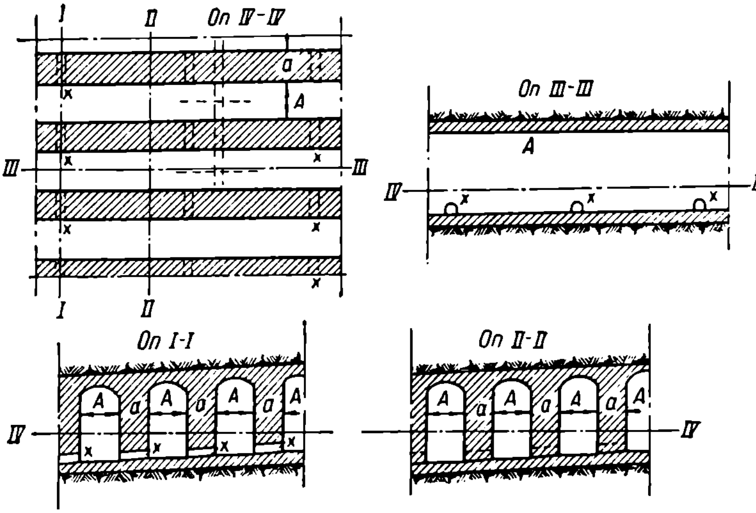



*Fig. 294.* Mining of rock salt with support pillars shaped like protective walls

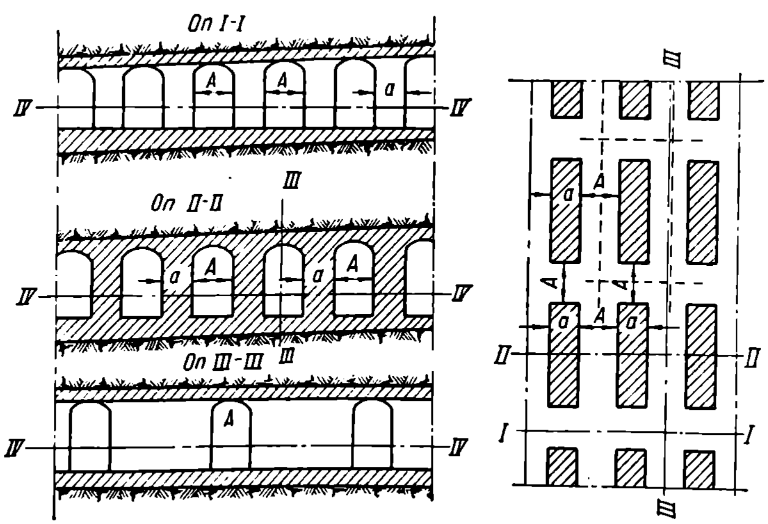



*Fig. 295.* Mining of rock salt with rectangular support pillars

but are liable to lead to the formation of fractures through which water from the overlying aquifers can penetrate into the mine. In certain conditions these fractures and cracks may even reach the surface and create openings through which surface water can get into mine workings, as is the case, for example, in nearly all coal mines and pits. Because of the ready solubility of rock (and potash) salt the appearance of water in the mine is fraught with grave danger. The size of the pillar and the width of the room (chamber) should be properly selected to prevent cavings. The pillars should be sufficiently big to bear the weight of all the superimposing rocks. There have been instances abroad of pillars of inadequate size leading to serious catastrophes. At the same time, to preclude unnecessary losses of the mineral, they should not be excessively strong or big. Computation of the size of the pillars based, on the one hand, on the estimated stresses they are exposed to and, on the other, on their strength, is discussed below (Section 4).

The maximal permissible width or span of rooms (chambers) is determined purely by experience since there is so far no reliable theoretic approach to this problem. Now that power undercutting (see below) has been introduced in development headings, better utilisation of the explosive power of charges requires a width of 17-25 metres, whereas formerly, when the operation was done by hand, it was 12-15 metres.

Depending on the disposition of rooms, pillars have the form either of protecting walls (interchamber pillars), separating neighbouring rooms (Fig. 294), or of rectangular support columns (Fig. 295). There are practically no square pillars in the Artyomovsk district. To connect adjacent rooms, the pillars are intersected every 30-60 metres by break-throughs 2-5 metres wide and 2-3 metres high, with vaulted or arched roofs.

The rooms are excavated in the rock salt only. In other words, it is necessary to leave a *protective ceiling* of salt (usually 1-3 metres thick) and a salt layer in the floor of approximately the same thickness. Quite often, to preserve the level nature of the floor in rooms extending along the dip or diagonally to the strike larger layers of salt, 10 and even 15 metres thick, unfortunately, have to be left in the bottom. The abandonment of salt masses in the bottom and roof is desirable because rock salt, from the standpoint of mining, is a very firm and compact rock. But if there is anhydrite (and not clay) above a salt bed, the roof of a room, particularly in a bed that is not too thick, is sometimes raised until it reaches anhydrite. It is made arched, and often quite low.

If beds dip insignificantly rooms are made to extend on strike and to the dip, or in oblique fashion.

In rooms lying on strike the floor is horizontal* (see Figs 294 and 295, section *III-III*), with the thickness of protective ceiling and bottom layers of salt as well as the height of the rooms themselves remaining uniform. The disadvantage of such a layout lies in the fact that break-throughs $x$, running across protecting walls and connecting the rooms, slope markedly, and that makes haulage of mine cars somewhat difficult. If the rooms were extended down the dip and the salt strata left in the roof and floor were to remain uniformly thick, they would have to be made to slope at the same angle as the dip, and this would make the rail transport inconvenient too. For this reason the floor of rooms in the whole of the mine field, or at least in most of it, is made horizontal, and that is why, with the thickness of the protective ceiling remaining stable, the height of the rooms and the thickness of the bottom layer gradually alter (see Fig. 295, sections *I-I* and *II-II*). Since the existing methods of mining do not provide for the recovery of salt left in the floor of the rooms, the above-described way of levelling out the bottom of rooms is regarded inefficient in spite of all its merits.

In view of this, and notwithstanding the handicaps referred to above, the arrangement of rooms along the strike is more advantageous. The diagonal position of rooms possesses features half-way between those of the two methods described earlier. When the rooms extend on strike, mining operations in the mine field progress schematically as follows (Fig. 296). Openings $AB$ and $AC$ are made along the dip, away from hoisting shaft $A$. The type of haulage for the transportation of salt in opening $AB$ depends on its gradient. Opening $AC$ runs sloping from the shaft. In order to reduce the track gradient and thus facilitate transportation, two diagonal haulageways $AB_1$ and $AB_2$ may be driven instead of one opening $AB$, whose angle of slope to the horizontal plane would be smaller. Ventilating shaft $D$ is usually near hoisting shaft $A$. A ventilation scheme with pillars in the shape of protective walls intersected by low break-throughs is shown in Fig. 296. Mining proceeds in all directions from the shafts of the mine more or less uniformly.

Let us now see how development work and stoping are done in the case of the Bryantsevsky bed, which, it may be recalled, is 40 metres deep and occurs with a slightly inclined dip at 150 metres from the surface. The system used in working this bed is shown in Fig. 297.

The width of rooms *1* is 17 metres, while the thickness of inter-chamber pillars *2* is 8 metres. The pillars are cut by break-throughs *3* every 30 metres, that is, their size at the bottom is $8 \times 30$

* Barring the slight slope given to the floor of the room to facilitate rail haulage.

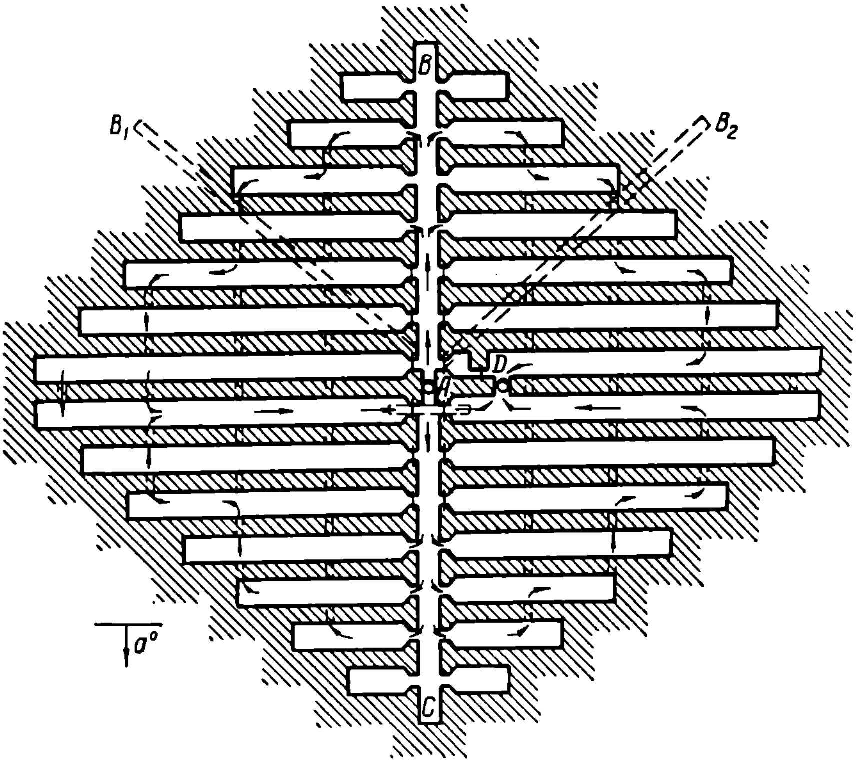


*Fig. 296.* Layout of workings in a salt mine field

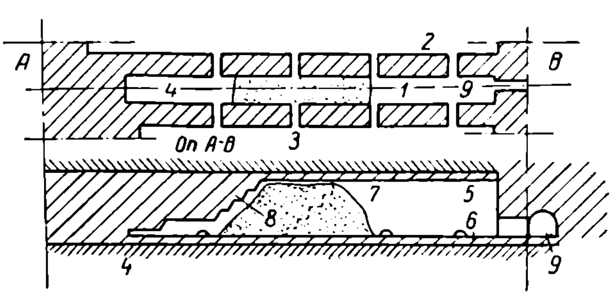



*Fig. 297.* Extraction of salt in a room

metres. The rooms are 25-27 metres high, since they have protective ceiling *5* and bottom *6* salt layers.

Before proceeding to stoping, *development openings* are made in each room.

Development opening *4* is run over the entire width of the room, but only to the height of 2 metres. The face is undercut with a coal cutter to a depth of 2 metres. A stope with an overall area of 32 sq m is undercut in one shift. Parallel with undercutting, 25-mm holes are drilled with electric augers. These holes are made in two staggered rows—one near the back of the working face and the other somewhat lower. The driller and his helper bore 300 metres of holes in a shift. Shooting is done with the aid of a bickford fuse. A blast in the development face yields about 150 tons of salt at a time. To transport it mine tracks are laid to the face.

Actual stoping starts after the heading of a development working has been advanced over the distance of two break-throughs from the "neck" of room *9*.

In order to make the room 25-27 metres high, the roof of the development working is first *underholed* near the initial site of the room, an operation sometimes termed "raising the roof". The face is broken so as to assume an overhand form. Going up gradually the face is enlarged in all directions. Drilling is effected from a *pile* of broken salt. To facilitate it, free space the height of man is maintained constantly between the breast of the face and broken salt. This is done by shovelling salt away.

When the face is brought to the planned height of the room, the pile of broken salt has two slopes, both with an angle of repose at 38-40°. The first *8* (anterior) faces the slope and the second *7* (posterior) is opposite the initial point of the room. Salt is further won from slope *8*. The pile of salt gradually grows in the direction of the room extraction, while the loading and delivery of the excavated salt are effected at slope *7*. Since salt broken from its solid mass increases in volume, a certain amount must be removed from anterior slope *8* too so as to have enough free room for drillers to work in and to provide passage for the ventilating air.

The height of benches or steps in a slope should be 1.5 metres. The monthly advance rate of room faces is of the order of 15-20 and more metres. The amount of explosives consumed in the mine is around 170-180 g per ton of salt mined.

From slope *7* salt is removed by scrapers. Each working room is serviced by two scraper hoists. They are set up on movable metal platforms 12-15 metres from the pile. They are operated by a team of six men: two at the hoists, two tending the scrapers themselves (chiefly breaking up large lumps of salt) and two—"slopemen"—on the pile. The team produces up to 600 tons of salt per shift. Salt at the front slope is loaded in the same way.

Since the breaking of rock salt requires a large amount of explosives and the working rooms are very high, the pattern of underground ventilation should provide for efficient *air circulation*.

Of interest in this respect is the ventilation scheme adopted at the above-mentioned Iletsk rock salt mine in the Urals.

The method of mining practised there (Fig. 298) is analogous to the one described above; the rooms are 25 metres high. The air enters the mine through hoisting shaft *1*, flows along haulageway *2* and then enters the rooms. Since airway *3* runs along interchamber pillars near the top of the rooms, the air current ascends via rise headings (break-throughs) *4* into airway *3* and is then cast up air shaft *5*.

The huge rock salt dome at the Iletsk mine, briefly described in Section 1, was worked for a long time, first by the open-cut method.

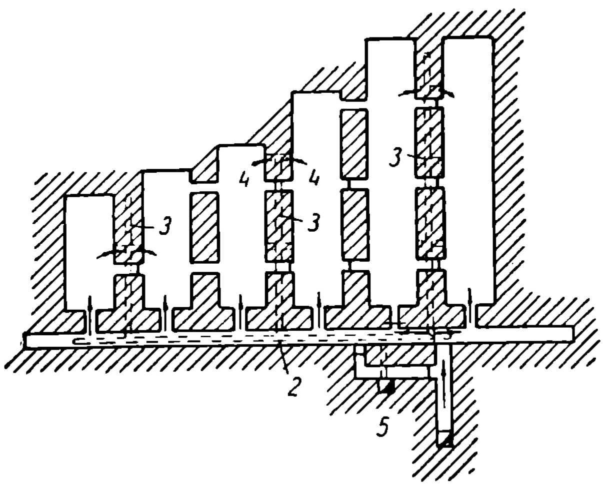



*Fig. 298.* Ventilation scheme at the Iletsk salt mine

From 1889 on rock salt was mined only in the so-called "old" room. In 1925 stoping operations there were suspended because there was a serious danger of the ceiling of the room caving in, caused by water leaking through cracks and dissolving salt. This abandoned room deserves mention because of its extraordinary size: it was 106 metres high, 14.5 metres wide at the top and 25 metres wide at the bottom, and 245 metres long. Inasmuch as salt was extracted by *underhand stopes*, the height (or rather the depth) of the room increased year in year out. The salt was hoisted up through a vertical "shaft", cut out in the wall of the room in the form of a vertical recess, which was deepened correspondingly. The room had no support, except for rafter timbering forming a canopy just under its ceiling.

When rock salt deposits are mined with support pillars of adequate size, the rooms, though of considerable dimensions, have no other artificial support. This makes it possible freely to use various machines for the sloping, loading and transportation of rock salt. In addition to ordinary cutting and loading machines and conveyers, extensive rooms may also be worked by small electric power shovels. Efficient operation of these shovels and mechanised loading equipment in general requires use of large-capacity mine cars, the size of the underground workings favouring this. Skip hoisting, however, requires setting up an underground coarse crusher plant.

## 3. Mining of Potash Salts

The Solikamsk deposit of potash salts is worked by mines extracting sylvinites and carnallites.

Occurring at the depth of 270-310 metres from the surface, the deposit has three workable beds: the Krasny II sylvinite bed—6-8 metres thick, AB sylvinite bed—2.5-3 metres thick, also called "particoloured" or "rich", and a carnallite bed, or rather, carnallite strata 70 to 100 metres thick. The pay beds are separated by intercalations of rock salt 3-6 metres thick and of sylvinite 1-1.5 metres thick.

As a rule, carnallites occur in the upper portion of the potash salt zone. It is usually capped by a layer of "cover" rock salt 40-70 metres thick which, in turn, is overlaid by an association of highly aquiferous argillaceous-marly rocks. Consequently, the cover bed of rock salt serves as a protective layer against the penetration of sweet water into the underlying strata of potash salts.

The deposit has a flat pitch with a regular general dip of 4-7°, but the rock series are very much plicated and, therefore, the bottom of the beds is distinguished by horsebacks.

The adopted method of mining is by rooms. The room axes are usually oriented in the direction of plication axes.

| | Sylvinite beds | | Carnallite |
|---|---|---|---|
| | AB | Krasny II | |
| Width . . . . . . . | 15-16 | 15-16 | 8 |
| Height . . . . . . . | 3-4 | 6-8 | 6-8 |
| Length . . . . . . . | 150 | 150-200 | 75 |
| Thickness of interchamber or rib pillars . . . . . . . . | 10-12 | 10-12 | 18-19 |

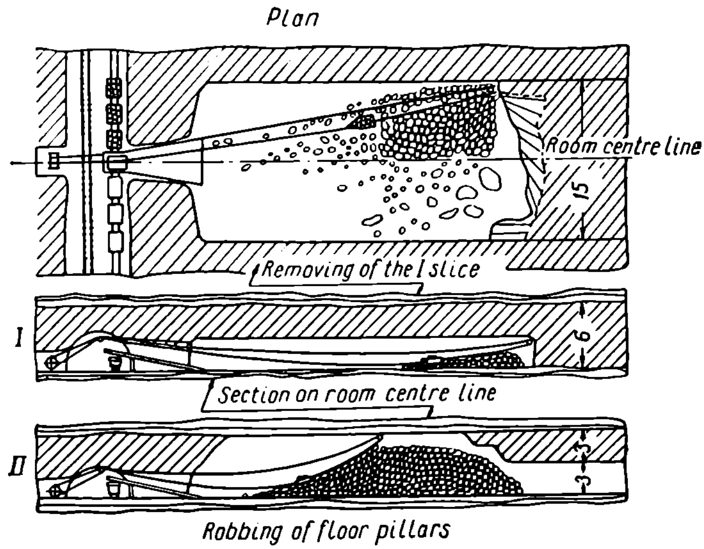



*Fig. 299.* Room mining of sylvinite

The reduced room span is adopted in *carnallites* because they are weaker and more water-adsorbing than sylvinites. The height of 6-8 metres adopted in the case of carnallites facilitates work in the rooms.

To make rib pillars stable, the axes of the rooms are exactly above one another in all the three beds.

When the sylvinite bed is 6-8 metres thick, its extraction is done in two slices, starting with the lower one (Fig. 299, *I*). The second slice is broken from a pile of blasted sylvinite (Fig. 299, *II*). As it accumulates, scrapers begin to haul the mineral towards the neck of the room. At the same time drillers in the interior of the room go on with ripping or slabbing the "ceiling". In this way the pile of sylvinite shifts from the neck of the room to its opposite end.

Carnallite is worked in the same manner, but from the top slice. This sequence of breaking carnallite is safer, for it makes it possible carefully to treat the roof of the room, to prevent it from caving in.

This method of mining is highly efficient: output per faceman per shift is as high as 25 tons, but it involves high losses of the mineral (about 50 per cent), chiefly in support pillars.

A different pattern of hole rounds is being tested to enhance the efficiency of drilling and blasting operations in the mining of potash salts. The new method implies driving a development opening 4 metres wide and 2.5 metres high along the room from which a fan-shaped round of long holes (locally called blast-holes) are drilled

(Fig. 300). The holes are made with electric augers with hard alloy bits.

Development openings are driven faster and at a lower cost by the ШБМ tunnel-boring machine (Fig. 301), with the aid of which a circular horizontal opening with a diameter of 3 metres is driven at a rate of 400-500 metres a month. The direction in which the machine

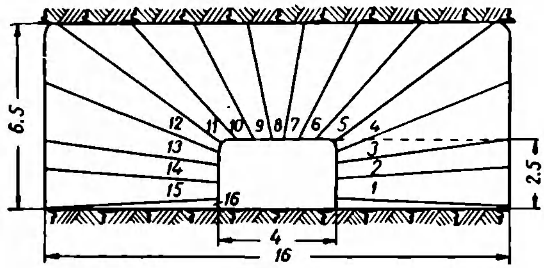



*Fig. 300.* Fan-shaped round of blast-holes

moves (see arrows in Fig. 302) is chosen conformably to the disposition of the rooms. Fig. 302 shows the position of rooms: *I* is in the stage of stoping with the mineral slushed by scraper hoist *1* to ore chute *3*; *II*—the room prepared for stoping; *III*—the room being prepared for stoping by tunnelling machine *4*; *IV*—worked-out room. Since the tunnelling machine is 6.25 metres long, special circular openings *2* have to be made on a radius of 5 metres to enable it to turn round near the ore chutes.

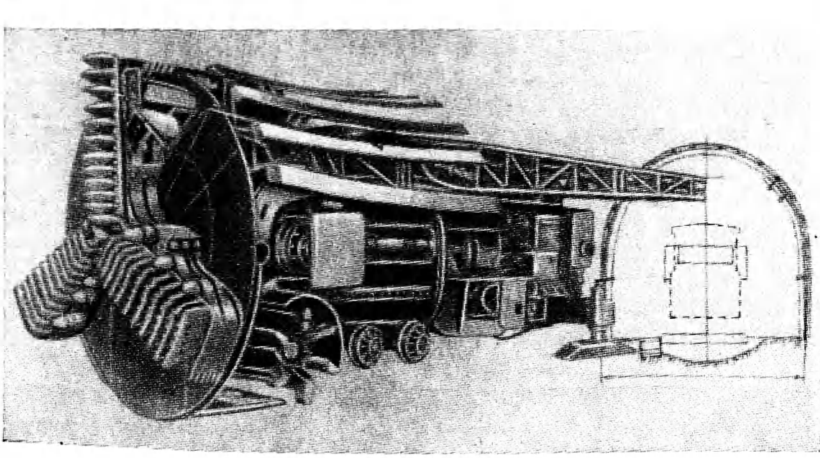


*Fig. 301.* ШБМ tunnelling machine

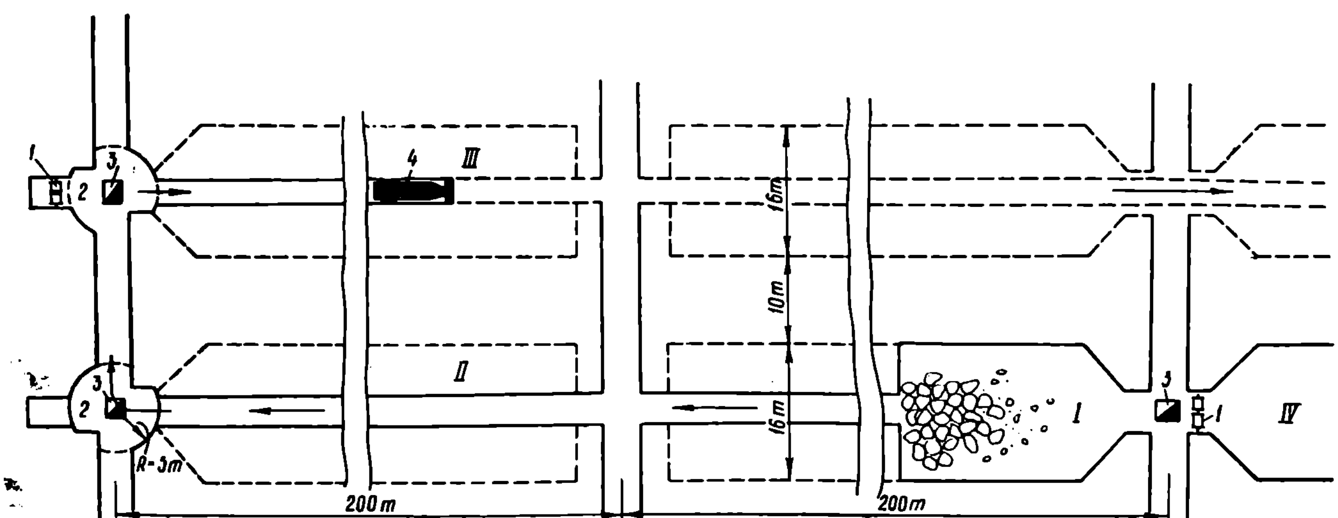



*Fig. 302.* Layout of rooms in mining sylvinite

In Germany potash salt deposits are invariably mined with *complete fill.* This is done because potash salts and, particularly, carnallite are not so strong and firm as rock salt. For this reason support pillars, especially if mining goes on at a great depth, are by themselves incapable of bearing massive rock pressure. Hence the purpose of abandoned pillars is here to maintain the stability of ground only during the actual extraction and stowing of a given room. Mass movement of ground after a large number of rooms have been worked out is prevented partly by the abandonment of interchamber support pillars, but chiefly by the complete filling of the goaf.

In mining potash salt deposits, prevention of rock from caving in into the worked-out areas also assumes particular importance because, due to the high solubility of salts, the penetration of water into salt workings results almost invariably in the complete destruction of the mine.

The filling materials are either the waste remaining after processing of potash salts on the surface or rock salt, both that obtained from various development openings driven outside the mined potash salt deposit, more often than not in the foot wall, and especially mined for filling purposes. Very frequently a mixed filling material is used—a combination of rock salt and tailings remaining after the chemical processing of potash salts. The filling can also be made by slushing (float-fill), but in that case saturated salt solutions (brines) are used instead of water. In the U.S.S.R. hydraulic (float) fill is employed at the Kalush potash salt mine.

The mining method applied varies, depending on the dip of a deposit. Fig. 303 outlines the method employed in working a *gently inclined* thick bed of potash salt. The development of a level provides for arranging slopes on strike every 200 metres. Protective pillars 5-6 metres thick are left on both sides of the slopes. These pillars are cut through by drifts run from the slopes every 20-25 metres. Outside the pillar such a drift is enlarged to the full width of the room (approximately 12 metres), with the height remaining at about 2 metres. This heading is run over the entire length of the room (around 60-80 metres), and after that the room is mined by the shrinkage method. As may be seen from Fig. 303, the floor of each room is flush with the level of the corresponding drift.

If a deposit is capped not by rock salt, but by anhydrite or saline clay, that is, by friable and readily breaking rocks, a protective mineral ceiling 1-4 metres thick is left in the roof of the room, as indicated in the figure. The cross-section of each room has then a characteristically trapezoidal shape. Between adjacent rooms on strike are support pillars which are later abandoned and, in addition to these, pillars 10-12 metres wide are left mid-way between two slopes. Triangular prisms of salt are also lost under each room.

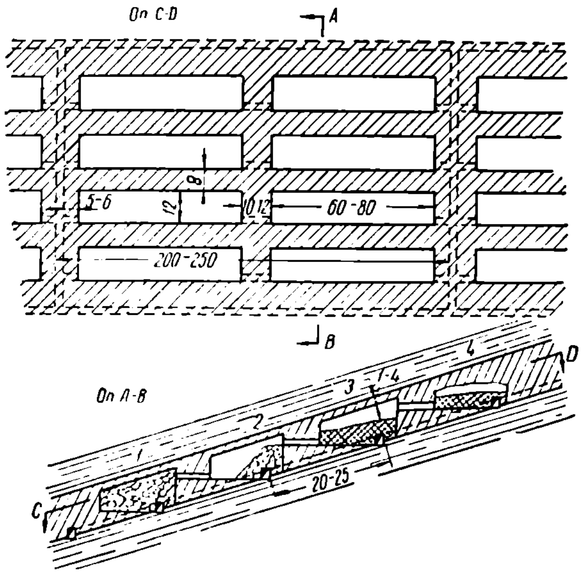



*Fig* *303*. Potash salt mining in a gently dipping deposit

The rooms are mined in the ascending order. When the stored salt has been removed from the rooms, they are filled. The filling material is brought down along the slope, first to the superjacent room, thence through a special rise heading in the pillar down to the one requiring filling.

The fill is put in place as compactly as possible and for that reason special stowing machines are sometimes used to bring it up under the roof. Fig. 303 shows room *1* already filled, room *2* in the stage of being filled, room *3*—broken salt is being hauled away, room *4* is in the stage of shrinkage-stoping.

Room mining in a steep bed is illustrated by Fig. 304. From shaft *1*, sunk in the foot wall of the deposit level, crosscuts *2* are run towards the bed. The vertical level interval varies from 30 to 75 metres. The levels are divided into sublevels *I*, *II*, *III*..., with vertical intervals of 6-10 metres. The sublevels are developed by short crosscuts *5*, *6*, *7*..., driven consecutively from blind shafts *4*, raised to the entire height of the level with certain intervals on strike. The sublevels are worked in the ascending order. The operations progress in the following sequence. Level drift *3* is run from main haulage crosscut *2* in the rock salt of the foot wall of the deposit, from which short crosscuts *5* are driven to the site of future rooms in the lower sublevel. When a crosscut like that is cut right into the midst of the deposit, a development opening about 2 metres high is first driven in salt over the entire length of future room *10*, extending all the way through the lateral thickness of the deposit (except for the protective ceiling of potash salt 1-4 metres thick sometimes left in the roof). Salt drawn from this opening is cleaned up, followed by the breaking and shrinkage-stoping of salt (as described above). The ultimate height of the room corresponds to the sublevel interval, that is, 7-9 metres.

Insofar as the other dimensions of the rooms are concerned, they depend on the thickness of the working deposit, the strength of salt and the firmness of wall rocks. Normally, the overall area of a room should not exceed 600-1,000 sq m with the more cavable and less viscid salts, but in more favourable conditions it can be increased to 1,500 sq m.

Support pillars *8*, measuring 12-15 metres, are left between rooms though more often 6-8 metres on strike. Cross headings (break-throughs) *9* are cut near the foot wall of a deposit to connect neighbouring rooms. Interchamber pillars in individual sublevels are immediately one over the other throughout the whole of the level interval. Sometimes reinforced (oversize) pillars up to 25 metres in length on strike are left every few rooms, which are separated by ordinary ones of the size indicated above.

When salt stored in shrinkage stopes has been withdrawn, the rooms are stowed with filling material. The latter, brought down from the surface or obtained underground, is delivered to the top level of a given room. Thus, for room *10* the fill is supplied via intermediary crosscut *6*.

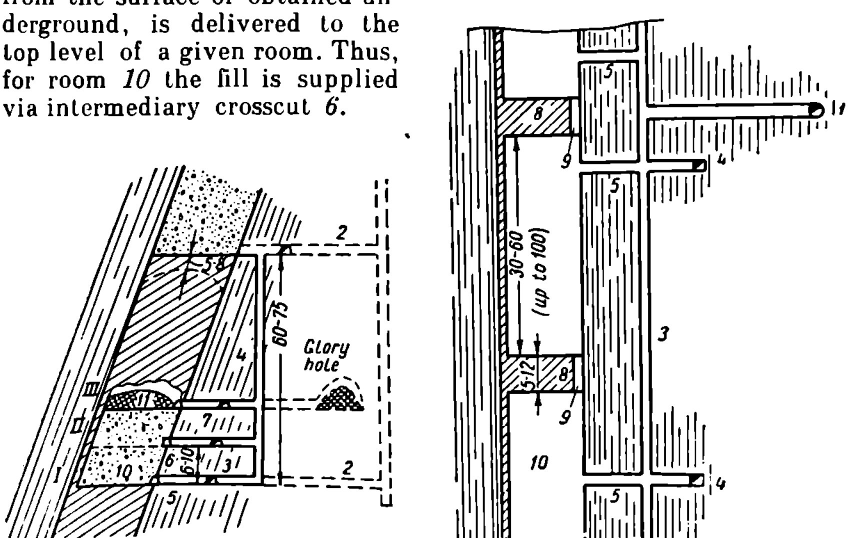



*Fig. 304.* Potash mining in a steeply dipping deposit

Rock salt, mined in glory holes, that is, in special chambers worked exclusively for mine-fill, is sometimes used as a filling material. This is illustrated in Fig. 304 (on the right side of the vertical section). It may be seen that the glory holes lie at the foot wall of a deposit, on a horizon overlying the sublevel room which is being filled. The interval between individual glory holes in one and the same sublevel is about 150 metres, while in adjacent sublevels along the dip they should not lie one over the other. The glory holes must not exceed 100 metres in length, 25 metres in width and 9 metres in height. Ordinarily, however, the actual dimensions are smaller.

The room to be filled is stowed so as to leave a free space about 2 metres high between its roof and the top of the fill. This space is for the drillers engaged in breaking salt in the superjacent room (room *II* in Fig. 304). The extraction of the overlying sublevels proceeds in similar manner, but salt is passed down to the main haulageway through intermediary drifts and slopes. Over the top sublevel room a pillar 5-8 metres high is left to serve as a sublevel floor pillar. The delivery of broken salt from the rooms and its loading into mine cars is usually done by a conveyer. The main strike drift is serviced by electric haulage.

In the case of very thick deposits, the above-described method of mining is modified: the stopes advance not along the strike but across it. In other words, the longitudinal axes of the rooms are turned 90° with respect to the line of strike.

The drawing of salt in rooms may also be effected from "subdrifts", in a manner analogous to that applied in mining ore bodies (see Chapter XXI, Section 3 below).

A specific modification of room mining is shown in Fig. 305. It is employed in working thick and medium-thick steeply dipping and inclined deposits. The underlying idea is to reduce the cost of haul-

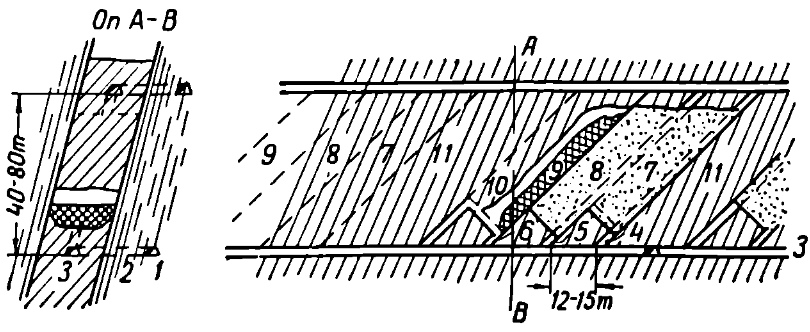



*Fig. 305.* Potash salt mining by inclined rooms

ing broken salt to the main level and to cut down the number of development openings by an *oblique* layout of rooms and inter-chamber pillars.

In this case a level with a vertical interval of 40 to 60 metres is not divided into sublevels. Crosscuts *2* are driven from main strike drift *1* in the foot wall of the deposit, and then merge into cross-drifts, from which strike drift *3* is made in the centre of the deposit. From the drift inclined raises *4*, *5*, *6* are put up every 12-15 metres. The first, after being carried 3-5 metres away from the strike drift, is extended over the entire thickness of the working bed, that is, across the full width of future room *7*. This inclined raise plays the role of a development opening, for it serves to undermine salt by firing explosive charges in its back. Its height should be sufficient to ensure efficient execution of this task. The raise must have a slope corresponding to the angle of repose proper of broken salt and filling material.

The inclined development opening is raised to the top level drift, but its last 5-8 metres are driven with a narrow face to provide for protective pillars under the drift. When this development opening is completed, inclined working *10* is cut out across the entire width and height of the future room at the bottom of the first, near the foot of its broader section. The undermining of salt in the room is then proceeded with. Salt is shrinkage-stoped in the room to the extent necessary to continue undermining the back from a pile of broken salt, and surplus salt is discharged periodically as the need arises down to the lower drift through an inclined slope. In this way the room is gradually worked out to its full height of 8-12 metres. The rooms are usually 60-100 metres long, depending on the level interval and their inclination. When salt in a given room has been extracted, the mineral is removed from it and the room is stowed with filling material supplied to it through the upper drift. Owing to the inclined position of the room, it is distributed throughout it by gravity. In this instance too the room is not filled to its roof. Sufficient free space is left between its inclined back and the top surface of the fill to permit initial undermining of salt within the bounds of the next room. Overlying neighbouring rooms *8* and *9* are worked in the same way as the first room, but the floor is made of mine-fill and not potash salt in situ.

To preclude massive subsidence of ground, inclined protective pillars *11* are left unrecovered every three-four rooms, their size depending on local conditions.

## 4. Estimating the Size of Support Pillars

The abandonment of support pillars entails considerable losses of valuable minerals and for this reason the method should be used for mining only the deposits whose resources in nature are practically inexhaustible, or those whose underground working it is imperative to protect from water inrushes at all costs. A typical combination of these two conditions is the mining of deposits containing rock and potash salts that dissolve readily in water. These salts are still mined everywhere by the method providing for the abandonment of support pillars (with the exception of low beds). In the U.S.S.R. all the major deposits of potash salts (Solikamsk, Western Ukraine) and rock salt (Artyomovsk district in the Donets coal fields, Sol-Iletsk, etc.) are worked according to this principle.

Whenever any given deposit is exploited by the method involving the complete abandonment of support pillars, determination of their size is a matter of prime importance. Excessive strength leads to unnecessary high losses of the useful mineral and tends to increase operative costs, for expenditure is charged against smaller recoverable tonnages. Insufficient rigidity of support pillars, on the other hand, is a source of numerous calamities, such as rock cavings, destruction of mine workings, surface plants and structures, and even the entire mine. Mining history in France and Germany has known catastrophes which sometimes acquired extremely large proportions, particularly in rock and potash salt pits.

A special method for estimating the size of support pillars in room mining was proposed a long time ago. Pillars are regarded as columns submitted to vertical loads corresponding to the entire weight of overlying rock, up to the surface (Fig. 306). This is, of course, a critical case, but it does occur when the total mining area is considerable compared to the depth of the mine.

In view of the immense thickness of salt beds and relatively small output of mines, the stoped-out area in a mine field increases rather slowly and rock pressure on the pillars therefore increases not over years, but over decades, and the size of pillars should be calculated beforehand so as to make them capable of withstanding this ultimate load. If this point is neglected, the pressure may crush them and destroy the whole mine.

Below is a series of formulas for computing the adequate size of support pillars (for details see the author's book *Fundamentals of the Theory of Coal-Mine Planning*).

Let us denote (Fig. 306) by:

$H$—distance between the top of a support pillar and the ground surface;

$h$—height of support pillar;

$s$—horizontal cross-section area of a support pillar (area in plan);
$S$—horizontal cross-section area of rocks per one support pillar (area in plan);
$q$—average unit weight of overlying rocks;
$q_1$—unit weight of support pillar rock;
$R$—compressive strength of support pillar rock;
$n$—margin of safety adopted in calculating the size of the support pillar.

Observations on the significance of the margin of safety or safety factor may be found below.

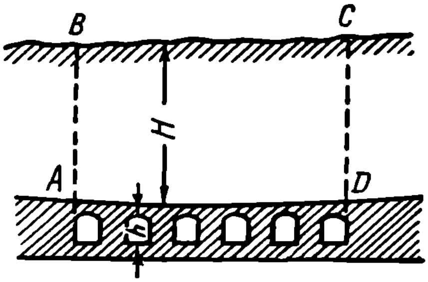



*Fig. 306.* Diagram for estimating the size of a support pillar

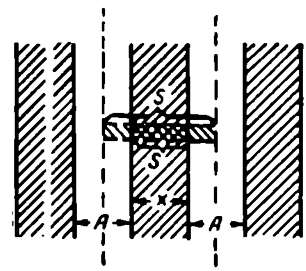



*Fig. 307.* Calculation of wall-shaped support pillars

Taking into account all that has been said before, we find that the conditions for the computation of the adequate size of a support pillar will be as follows:

$$SHq + shq_1 \leqslant \frac{sR}{n}. \tag{1}$$

Hence, for the critical case it is

$$\frac{S}{s} = \frac{R}{nHq} - \frac{hq_1}{Hq}. \tag{2}$$

The algebraic form of the proportion $\frac{S}{s}$ depends on the configuration of the horizontal section of support pillars and that of their circumjacent mine workings (in particular, rooms).

Let $A$ be the width or span of a room, $x$—the width (that is, the shorter side of the rectangular area in plan) of a support pillar.

From (2) we obtain the following formulas for computing the width of support pillars:

1. Pillars in the shape of walls (Fig. 307):

$$\frac{S}{s} = \frac{A+x}{x},$$

hence
$$x=\frac{A}{\frac{R}{nHq}-\frac{hq_1}{Hq}-1}. \tag{3}$$

2. Square pillars:
$$\frac{S}{s}=\frac{(A+x)^2}{x^2},$$

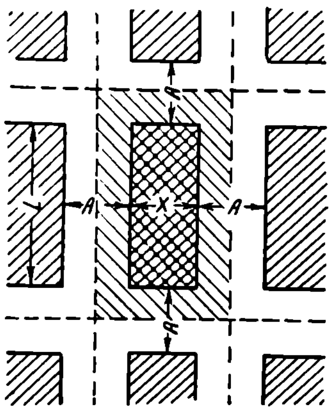


*Fig. 308.* Calculation of constant-length support pillars

and accordingly,
$$x=\frac{A}{\sqrt{\frac{R}{nHq}-\frac{hq_1}{Hq}-1}}. \tag{4}$$

3. Pillars have a length of $L$ (Fig. 308):
$$\frac{S}{s}=\frac{(A+x)(A+L)}{xL}$$
$$x=\frac{\frac{A^2}{L}+A}{\frac{R}{nHq}-\frac{hq_1}{Hq}-\frac{A}{L}-1}. \tag{5}$$

4. Pillars with the proportion of $\frac{x}{Z}=c=\text{const}$; since here
$$\frac{S}{s}=\frac{(A+x)\left(A+\frac{x}{c}\right)}{\frac{x^2}{c}}$$

then
$$x=\frac{2Ac}{-c-1+\sqrt{(c-1)^2+4c\left(\frac{R}{nHq}-\frac{hq_1}{Hq}\right)}}. \tag{6}$$

5. Pillars with the length of $L$ are surrounded by rooms of unequal width $A$ and $B$ (Fig. 309):
$$\frac{S}{s}=\frac{(A+x)(B+L)}{xL}.$$

Accordingly,
$$x=\frac{A+\frac{AB}{L}}{\frac{R}{nHq}-\frac{hq_1}{Hq}-\frac{B}{L}-1}. \tag{7}$$

In the formulas (3)-(7) it may be assumed that $h=0$, if the height of pillars is insignificant compared to the depth of mining.

In the above-mentioned formulas the width of rooms is taken to be predetermined. So far there are no reliable methods of determining the width of rooms and this is established in a purely empirical way, most often within the range of 10-17 and less frequently 23-25 metres (Sol-Iletsk mine).

Let us make some remarks on the significance of the values of the above formulas. Compression strength $R$ of the material contained in the support pillar is established by the laboratory tests of rock specimens under a special press.

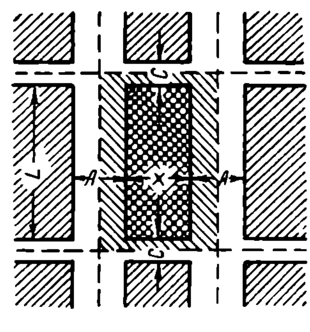


*Fig. 309.* Diagram for estimating the size of support pillars

It should be noted that compression strength (in kg/cm²) depends on the absolute size and shape of the tested specimens. For instance, if it is cubes with edges 5, 10, 15 and 20 cm long that are tested under a press, it becomes evident that the value of $R$ for the same material is apt to rise. Therefore, it is better to take cubes whose edges are not less than 15-20 cm long.

Still more conspicuous is the effect of the *shape* of specimens. For prismatic specimens the greater the ratio of $\frac{h}{a}$ ($h$ — height of the specimen and $a$ the length of the edge in the square base) the smaller $R$. In the introduction of correction factors for the shape of specimens the following data may be accepted as guidance:

| | | | | |
|---|---|---|---|---|
| Ratio $\frac{h}{a}$ . . . . . | 1 | 2 | 3 | 4 |
| Correction factor | 1 | 0.8-0.7 | 0.6-0.5 | 0.5-0.4 |

In using the formulas (3)-(7), it is necessary, after the determination of the size of the pillar and the establishment of the ratio $\frac{h}{a}$, to effect, when needed, a conversion by introducing a correction factor for the shape of the specimen. A. Penkov and A. Vopilkin have proposed a grapho-analytical method of calculating the size of support pillars with due account of their shape.

The value of compression strength (in kg/cm²) for the cubic specimens of rock salt is about 300-450, for sylvinite 270-360, and for carnallite 60-160. It should be noted, however, that the compression strength of a mineral should not be adopted from handbooks in making concrete estimates, but established by laboratory tests because the strength properties of rocks may be quite variable.

Considering that the loads pressing on support pillars are completely static in nature, the safety factor $n$ may be adopted at its minimal value—2.5-3.0.

The average unit weight of cover rocks (capping) may be taken at 2.3-2.5 $t/m^3$. The unit weight ($q_1$) of rock salt is about 2.2 $t/m^3$ and that of potash salts 2.1-2.2 $t/m^3$.

Formulas (1)-(7) for flat deposits also hold good for the gently inclined. The above-mentioned conditions are applicable to all the major deposits in the U.S.S.R. worked by the method of abandoning support pillars and, therefore, any special discussion of instances involving deposits with a sloping and high dips, though desirable, is nevertheless of a lesser interest.

If the depth value $H$ changes along with the dip of the deposit or the uneven nature of the ground surface, the formulas (3)-(7) may be applied to individual sections of the mine fields in which the value $H$ may be considered to be approximately uniform.

When the deposits occur at a considerable depth or the useful mineral is weak or the ratio $\frac{S}{s}$ actually adopted is too high, mining with support pillars alone is either impossible or, because of large losses of the valuable mineral, irrational. Critical depth $H$ may be found from the inequation (1). Support pillars have to be reinforced by filling from this depth.

## 5. A Few Remarks on the Production of Rock Salt by Dissolution

The capacity of rock salt readily to dissolve in water may be utilised for its mining.

The boreholes drilled from the surface down to the rock salt bed are lined with casing pipes throughout the strata of rocks capping the bed. Another set of internal inlet pipes is inserted in those pipes to feed the water dissolving the salt, while the brine ascends to the surface via an annular clearance space between the two sets of pipes and is then delivered for processing to chemical, usually soda, plants.

Despite its simplicity, this method has serious drawbacks: the percentage recovery of salt reserves from a deposit does not exceed 5-10; leaching is very difficult to control and apt to cause breakdowns; the brine is contaminated by other admixtures; the dissolution of salt over large areas may cause sinks and pits on the surface, etc.

For these reasons it is better to use the system of mining by shaft openings, that is, underground leaching of rock salt in rooms.

## 6. Mining of Building Stones by Underground Methods

Besides rock and potash salts, the method of room mining with abandoned pillars of the mineral is used for working limestone, gypsum, roofing slate and other building material deposits.

Around Odessa and in many parts of the Crimea there are large deposits of Tertiary *limestone* which is a building material widely used in urban construction in the south of the U.S.S.R. One of the varieties of this limestone—coquina — is so soft it can easily be sawn by both hand and power saws (Fig. 310). Occurring flatly, coquina beds are mined predominantly from valley slopes through adits and a network of rooms ("galleries") 4-5 metres wide with rib pillars in-between.

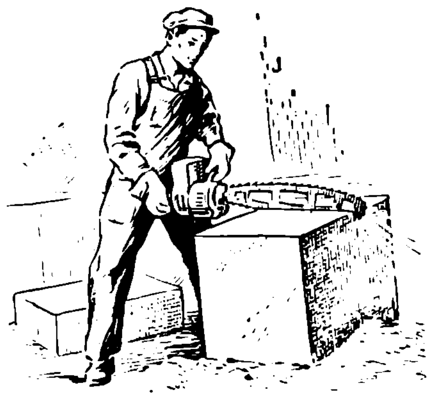


*Fig. 310.* Sawing a coquina block with a power saw

The stone in rooms is detached first from the breast of the face in large blocks, the size made to conform to that required for the ashlar. To protect blocks from breaking, brushwood is laid out at the site of their fall. Special Victor-Ragozinsky machines are employed for winning coquina blocks. After their extraction, they are sawn up into $51 \times 25 \times 21.5$-cm pieces. For this work *electric saws* of the types applied in the timber industry are now widely used. They are driven by 1.3-1.6-kw electric motors. The width of the cut is 9 mm and the overall weight of the saw—17 kg. Coquinas can be sawn with electric saws both in underground and open-cut mining. The sawn stones are stacked up on the surface, where they gradually dry and become markedly harder.

Fig. 311 is illustrative of the room mining method applied in working a thick steeply dipping deposit of *roofing slate*. It is extracted by overhand stoping which makes it possible to obtain slabs of considerable length and width. The usual size of rooms: length on strike—up to 30 metres; that across the strike should conform to the lateral thickness of the deposit; height—up to 20-35 metres. To make the roof of the room sufficiently stable, it is made arched.

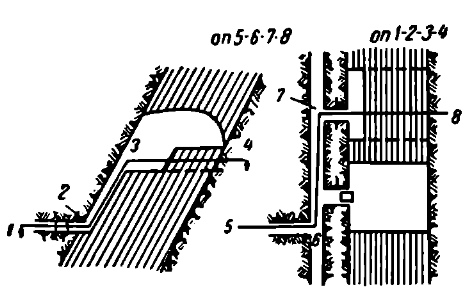



*Fig. 311.* Room mining of roofing slate

# *ORE DEPOSITS*

CHAPTER XIX

## CHOICE OF METHODS FOR MINING ORE DEPOSITS

### 1. Preliminary Remarks

General concepts relative to the methods employed in mining valuable minerals and the basic requirements an appropriate working system should meet concerning safety, low costs and minimal losses of the mineral, have already been discussed in Chapter VII.

Below we shall dwell briefly on the properties of ore deposits which are of technical significance in choosing the appropriate methods of mining.

The geological structure and texture as well as the matter composition of ore bodies are extremely manifold due to the diversity of their *genesis* and subsequent *tectonic* phenomena.

The external *shape* of ore bodies has already been discussed in Chapter III, which described the methods employed for their opening. We have seen that by their external shape ore deposits may be classified into *beds*, *sheet* or *blanket-like deposits*, *placers*, *lenses*, *veins* and *lodes* and *ore bodies of irregular outline.*

### 2. Thickness of Ore Occurrences

The thickness of ore deposits varies extremely widely—from a few centimetres and even millimetres (for instance, cinnabar veins) to hundreds of metres. The thickness of a deposit is one of prime factors borne in mind in selecting the proper method of mining. Both in the instance of beds and of ore bodies of other shapes there may be a *true thickness*, that is, measured along the normal line to the foot and hanging walls, and *horizontal*, or *lateral*, measured from the hanging wall to the foot in the horizontal plane. Inasmuch as ore deposits are quite frequently characterised by variations in thickness over short distances, the definite values of true and lateral thickness refer mostly to individual parts of the ore body, or else denote its *average* or *mean* thickness. In the latter case, *minimum* and *maximum* thickness should also be indicated.

From the standpoint of mining, it is customary to classify ore deposits into the following groups:

I—very thin (less than 0.7-0.8 metre);
II—thin (from 0.7-0.8 to 2 metres);
III—medium-thick (from 2 to 5 metres);
IV—thick (from 5 to 15-20 metres);
V—very thick (over 15-20 metres).

In his book *Mining of Ore Deposits* (1954), M. Agoshkov substantiates this classification as follows:

1. *Very thin* deposits include those where the driving of development openings and stoping proceed simultaneously with the blasting of the enclosing country rocks.
2. *Thin* deposits are those where stoping can be done without the blasting of enclosing rocks, though the drivage of lateral development openings requires blasting most of the time.
3. In *medium-thick* deposits the blasting of enclosing rocks is not done either in stoping or in development work. The maximum thickness for the use of stull timbering in these deposits is 5 metres.
4. *Thick* deposits include those where stoping in high dip can be done over the entire thickness of the ore body on strike.
5. *Very thick* deposits are the ore bodies measuring over 15-20 metres. In stoping they are separated or cut into blocks along their thickness or else extracted across the strike.

The classification of ore deposits according to their thickness should not be too rigidly adhered to, for many ore bodies are apt to vary in thickness over short distances.

To form a proper judgment of the size of an ore body, especially one with an irregular outline and high dip, one should consider its *area*, that is, the area of the horizontal section of an ore body in any of its levels.

## 3. Angle of Dip

For genetic and tectonic reasons, ore bodies or individual parts can occur at greatly varying angles of dip. Hence there are *flat* or *flatly sloping* deposits with an angle of dip of up to 30° (particular attention in this group should be paid to horizontally occurring deposits); *inclined* or *sloping* deposits with an angle of dip of 30-45°, and *steep* or *high-dipping* (45-90°) deposits.

Most ore deposits are distinguished by a steep dip. This applies especially to veins on account of the nature of their geological origin. Tectonic cracks and fissures, marking the lines along which the disruption and shifting of the ground occurred, usually, though not always, assumed a vertical or steep position in the earth crust. Correspondingly steep, therefore, are the veins which originated when these cracks were filled with ore matter. True, sometimes flat veins and underlying lodes have to be mined too. This is due, firstly, to the fact that the veins arising sometimes were not steep and, secondly, to the fact that the old veins, originally nearly vertical in space, as is characteristic of veins in general, became inclined or flatly dipping in the course of subsequent tectonic displacements.

The biggest Soviet deposits of iron ore, worked by the underground method (Krivoi Rog, Vysokaya Mountain in the Urals, Gornaya Shoriya), occur at steep angles of dip. Cupreous pyrite lenses in the Urals are as a rule steeply dipping ore bodies too.

On the other hand, gold placers, some copper ore deposits (for example, Dzhezkazgan in Kazakhstan), iron ores (central regions of the U.S.S.R.) and the largest manganese ore deposits in the Nikopol and Chiaturi districts, are horizontal or rather flat.

Typically sloping are the bauxite deposits in the Northern Urals and a number of ore deposits in Altai and Kazakhstan.

The angle of dip of some ore bodies varies substantially over short distances.

## 4. Depth of Ore Occurrences

The depth of ore deposits is extremely manifold. There are deposits which, because of their geological origin, are very shallow-seated. One example is the ore bodies whose formation is linked with alterations (weathering) of rocks and minerals near the surface. Thus limonite or brown iron ore deposits sometimes appear near the outcrops of limestone; suitable for pig-iron smelting, but being nothing but surface formations, they vanish at a depth of a few scores of metres. Gold placers formed in the existing river valleys are also shallow-seated. If the ore deposit of a sedimentary type occurs in rocks of an older age but lies in an area which has not been subjected to any considerable tectonic movements of the earth crust, it may be shallow-seated too (for example, manganese ores in the Nikopol district). But since the sedimentary deposits and ore accumulations (bodies) in the residium (and placers) formed in the remote geological ages and were later subjected, together with the entire set of rocks enclosing them, to orogenic processes, they too may now be found occurring at great depths. A striking example are the iron ore deposits of Krivoi Rog which, as it has now been established, were originally of a sedimentary nature, but were gradually subjected to metamorphism and tectonic influences and, occurring steeply, sank to a depth so far not determined by geological investigations. Nor is there any definite information as to the maximal depth of the steeply dipping deposits of cupreous pyrites in the Urals.

Also very deep-seated are most of the lode deposits. The prospecting data available at present and geological considerations suggest that the gold strike reefs of the Berezovsky deposit (in the Urals) occur at considerable depths.

It should be borne in mind that if some veins or ore bodies in general in a deposit do pinch out at a greater depth there may be other, *blind* ore bodies, which do not reach the surface.

## 5. Matter Composition of Ores

Sometimes the metals contained in a deposit are in a *native* or *natural* state, such as gold, platinum, platinoids (for instance, osmiridium), at times copper. Ordinarily, however, the ore comprises metals combined chemically with other elements.

There are *simple* ores—containing but one metal and complex (*polymetallic*) ones, when one and the same ore carries two or several metals in quantities making them suitable for commercial utilisation.

Simple ores are encountered much more rarely than those containing two or several metals. Major iron ores (magnetite, hematite, limonite) are generally iron oxides, but even these sometimes carry other valuable constituents (for example, titanomagnetites containing vanadium). Among the typical polymetallic ores are galenoplumbic and argento-plumbic ones, cupreous pyrites of the Urals, which often contain, in addition to copper, zinc, gold and some trace elements.

Many nonferrous ores are distinguished by the fact that they are *sulphides*, that is, sulphurous compounds.

There have been instances of technological progress causing the revision and reassessment of the industrial importance of ores. *Bauxites*, now used as a raw material for alumina for the subsequent manufacture of aluminium by electrolysis, were in the past considered lean iron ores. Carnallite, which some time ago was regarded merely as a raw material for fertiliser, has now become one of the ores from which magnesium and magnesite compounds are derived.

The proportion of valuable components rendering the ore suitable for commercial utilisation depends on the actual state of technology and economic conditions and, therefore, cannot be expressed in definitely set figures. To give a rough idea, let us quote the *Geological and Prospecting Glossary.*

"The approximate minimal content of metal in an ore allowing its commercial utilisation is as follows: 1) iron ores—ferrum 30 per cent; 2) copper ores—copper 0.5-0.7 per cent; 3) plumbic ores—lead 2-5 per cent; 4) zinc ores —zinc 20-25 per cent; 5) gold ores—gold in primary deposits 3 g per ton, in placers 0.1-0.15 g per ton;

6) mercury ores—mercury 0.5-1 per cent; 7) cassiterite—proportion in placers 0.1-0.5 per cent. Due account should be taken, however, of the relative nature of the figures listed, which may vary depending on the complex set of present-day economic conditions."

From the standpoint of mining, the changes which the matter composition of ores undergoes near the surface, in the weathering zone, are of importance. Superimposing the occurrences of hematite and magnetite are aggregations of limonite. The *gossans* originating over iron-bearing sulphide deposits contain concentrations of metals that are found but in small amounts in unaltered (*primary*) ores. The gossans overlying cupreous pyrite lenses in the Urals, for example, present an elevated proportion of gold, and copper is no longer contained in sulphides, but is found in the form of oxides.

Extraction of metals (copper, nickel, cobalt, molybdenum, etc.) from normal (nonoxidised) sulphide ores is as a rule done by flotation, but to treat *oxidised* sulphides other concentration and processing methods are necessary. Consequently, to secure successful concentration one should establish a clear-cut demarcation line at depth between oxidised and nonoxidised ores during the mining of the deposit and to make provisions for their separate extraction and delivery from the mine.

Since the oxidation of sulphides begins with the appearance of very thin films on the surface of ore lumps which, however, may impede the process of flotation, to keep sulphide ores long during their shrinkage-stoping can unfavourably affect all the subsequent dressing operations.

Of prime importance for mining operations is the capacity of some sulphide ores to *grow hot* and *self-ignite*. This phenomenon has many features in common with the spontaneous combustion of coal. Underground fires caused by the spontaneous combustion of sulphide ores break out when mining entails substantial ore losses, when the abandoned ore has been broken, and its aggregations are exposed to outside air. These conditions arise in working sulphide ores with caving. The presence of timber in the abandoned ore masses increases the hazard of self-ignition. The best *prophylactic* measure against underground fires in mines with spontaneously igniting ores is mining with *complete filling*. Underground fires caused by the spontaneous combustion of ore are controlled by sealing off of fire-stricken sections (building of fire-breaks) with subsequent *silting*.

Spontaneous combustion of ores is a feature of great importance, particularly in mining cupreous pyrite deposits in the Urals.

*Contacts*, that is, interfaces between the ore body and the enclosing barren rocks, may be *sharp* (*distinct*) or indistinct (*poorly defined*). In the latter case, the regular *assaying* of enclosing rocks has to be effected in the stopes to determine the proportion of the useful components they contain and to avoid their loss. Sometimes valuable metalliferous components are *disseminated* in the enclosing rocks—small and minute inclusions whose presence can be established only by chemical analysis or by the examination of mineral sections under microscope. Such minute disseminated inclusions of ore in country rocks are known as *phenocrysts* or *impregnation ores.*

The useful constituents may be distributed in an ore body quite irregularly. If there are any specially enriched portions or sections of a vein or an ore body in general, they have to be extracted and taken out of the stope *separately* (selectively). The term *selective mining* is also applied to cases where two or several useful minerals occur together in the deposit and each is extracted separately from the others. Finally, in mining very thin veins or beds, it is necessary to blast the enclosing country rocks to obtain an active stope area of sufficiently large size. If the useful mineral and barren rocks are drawn and taken out from the stope without any preliminary separation, this is designated as *bulk* or *wholesome* mining. If, on the other hand, the valuable mineral is broken and hauled from the barren rock separately, the stoping is said to be *selective.* In the latter case, separate stoping requires sharply marked contact and the ability of the mineral to become readily segregated from the enclosing rocks.

The admixture of barren rock to the ore during the stoping process results in *dilution* or *contamination* of the ore, and that is deleterious to mining operations in general. The significance of dilution is to be viewed in a somewhat different light when wall rocks include impregnation ores. Dilution is then attended by an increase in the total amount of useful constituents through the addition of phenocrysts. The maximal level of wall rock admixtures is in such cases determined by technical and economical estimates, with due consideration of both mining and concentration operations.

The *value* of useful components in the ore expressed in money per unit weight of concentrates and metals recovered may vary considerably.

## 6. Hardness and Strength of Ores and Enclosing Rocks

The hardness of ores and wall rocks may be assessed with the aid of the special scales elaborated by M. Protodyakonov. The scales used at present to evaluate the drillability of ores and rocks are based on drilling rates. They have been established for individual mining enterprises and even entire areas (for instance, Krivoi Rog and the Urals). Since, with few exceptions, ores and enclosing country rocks in ore deposits have to be blasted out because of their hardness, proper evaluation of their hardness is a matter of prime importance in mining ore bodies.

Of equal importance is adequate evaluation of the degree of *rigidity* or *stability* of ores and wall rocks. This notion refers not only to the hardness of the valuable mineral or the enclosing rocks but to their jointing and ability to exfoliate, as well as to their moisture content. In general, rigidity (stability) means the capacity of ores or rocks to stand firm without caving in spontaneously as the result of exposure from below or on the sides. Such a definition of rigidity is of course incomplete and insufficient for practical use because it fails to indicate the *degree* of rigidity. To bring some clarity into the issue of quantitative determination, there have been suggestions to assess the extent of rigidity or firmness by the maximal area of exposure from below, under which the rock retains equilibrium and does not cave in. Such a definition, however, would be defective, for firmness depends not only on the size but also on the configuration of the exposure area, and particularly on the ratio of its length to its width. For these reasons one has to content oneself with a *qualitative* appraisal of the rigidity of rocks and valuable minerals.

M. Agoshkov treats this issue as follows:

"The choice of mining method and appropriate mode of supporting worked-out areas requires the following classification of rocks in accordance with their stability (rigidity):

"1. *Very unstable* ground allowing no exposure of the back and walls of a mine working without support. Advance timbering of the roof and sometimes of the walls is imperative in driving openings in this ground. In working ore deposits, one rarely comes across friable, loose and running, and water-saturated ground.

"2. *Unstable* ground, permitting small areas of exposure in the back and walls and requiring strong support to be set up immediately after stoping; it occurs more frequently than rocks of the first group.

"3. Rocks of *medium stability*, allowing us to leave large areas of exposure with no support following stoping operations. The need to support this ground arises with time, not at once.

"4. *Stable* rocks, permitting very extensive areas of exposure to be left both from below and on the sides, without any support. The rocks of the third and fourth categories are encountered in ore deposits most.

"5. *Very stable* ground, permitting a very large area of exposure both from below and from the sides without any support. It is encountered in the mining of ore deposits somewhat less than the preceding two categories.

"The nature of caving is of prime importance for making a proper appraisal of ground firmness in the choice of the mining method: does it occur suddenly and instantaneously over an extensive area, or gradually, covering small sections; in relatively small lumps and layers or in large blocks; can its onset and extent be predicted by outward signs, or not, etc.?

"Quite often the ground does not reveal any signs of instability immediately after its exposure, but becomes weak and starts rushing in in lumps or large blocks at some later date under the impact of atmosphere and water; occasionally, with time, it develops a tendency to swell or bulge." (M. Agoshkov, *Mining of Ore Deposits*, 1954.)

The degree of stability of any given ground or ore plays a major part in the selection of the mode of supporting mined-out space and the evaluation of the possibility of the ore being diluted during stoping operations.

It should be noted that the possibility and nature of ore and ground caving must be viewed not merely from the standpoint of the harm they bring to mining and of elaborating adequate measures to forestall this harm, but in some cases also of *profitably* utilising the property of rocks and ore to cave or break. Thus, in Chapter XXI below, we shall have an opportunity to discuss a system of mining thick deposits by horizontal slicing with the caving of cover rocks, its successful application depending, in particular, on the ability of the ground to cave in rapidly and regularly right after the ore has been extracted. Furthermore, there are highly efficient methods of mining involving the block caving of the ore whose application presupposes the breakage of the ore into small pieces in the process of caving.

The ore should be blasted in such a way as to prevent its pieces and lumps from being stuck in the discharge openings of chutes when they are loaded or in the goaf during the shrinkage-stoping process. The cross-section of discharge openings in the ore chutes must be 4-5 times as large as the biggest of the ore lumps.

Large blocks of ore undergo *secondary breaking* or *block-holing* to make it possible to load them into mine cars.

Some ores, containing clay or a high proportion of fines, especially moist ones, are liable to *compact* when stored in stopes. This property should be taken into account in choosing mining methods, for it may present an obstacle to the employment of shrinkage stoping.

## 7. Size Grading of the Ore

Size grading of the ore denotes its granulometric composition, that is, quantitative proportions of the various lump sizes it contains.

The specifications set up by modern industry concerning the size grading of valuable minerals are rather stringent. Permissible percentages of fines and oversize (coarse fractions) are fixed for ores supplied by individual mines and basins, with a limit placed on the maximum size. According to market specifications, in the case of Bakal iron ores the proportion of fines up to 3 mm in size should not exceed 5-8 per cent; the maximum permissible size of a lump (the so-called marketable piece) for the same ores is 300 mm. An increased content of fines entails a sharp rise in fuel consumption in blast-furnace smelting and tends substantially to lower furnace efficiency. This also harmfully affects the smelting of many nonferrous ores. Moreover, the augmented yield of undersize increases the amount of metal waste during the mining and transportation of the ore, since the finest ore fractions, the ones most likely to be lost, as a rule contain the highest percentage of the metal. Equally undesirable are the unnecessarily large ore pieces, for they cause serious difficulties in mining and impair labour efficiency during the extraction of the ore and its subsequent processing at the concentration mills. In modern underground mines the size of the marketable piece varies widely—from 200 to 900 mm and even one metre. In most of the mines the size ranges from 250 to 400 mm. Lumps of considerable size are encountered in large mines employing mass-production methods. The trend in recent decades has been towards augmenting the size of the marketable piece, for this permits eliminating labour-consuming block-holing operations. The drawing of ore of larger size, however, requires the installation of especially strong and well-equipped ore chutes, powerful haulage facilities, large-size crushers, and that is justified economically only in large-scale mining.

The capacity of the ore to break into pieces of different size following its detachment from the solid mass should be taken into account in selecting mining methods. It has been observed, for example, that in the process of chuting the smaller pieces filter through the larger ones and come out faster through the discharge opening. Therefore, if the overlying ground breaks into smaller pieces than the ore during the spontaneous breakage, it is not advisable to apply the spontaneous-caving methods. In this instance, it is necessary artificially to break the ore to obtain sufficiently small pieces.

The main and most efficient means of obtaining the desired size grading of the ore is proper adjustment and control of the parameters accepted for drilling and blasting operations in stoping. The basic factors influencing the size grading of the ore or barren rock obtained by blasting are:

1) physical and mechanical properties and, above all, jointing and degree of disruption of the solid ore or ground;

2) method of breaking. The best (uniformly fine) breakage is ensured by heavy blasting (in holes or long blast-holes), and the worst by coyote blasting, when the ore or ground in the sections immediately bordering on the charge is ground to powder, while that farther away is detached in large blocks;

3) the undercut area and the number of exposed surfaces in the solid block subject to blasting. The larger this area and the greater the number of these surfaces, the higher—all other conditions being equal—the yield of the oversize;

4) total size, division and uniform distribution of explosive charges in the block subject to blasting. The more dispersed and uniformly distributed the given quantity of explosives in the block, the

smaller is the size of broken ore, provided the amount of each individual charge is sufficient to ensure normal detonation;

5) brisance of explosives, sequence of their firing (instantaneous or consecutive) and proper conduct of blasting operations.

## 8. Some Other Factors Affecting the Choice of Mining Method

In most ore deposits the *abundance of water* plays no particular role in the choice of mining method. But in some instances it is a factor of considerable significance. In mining placers, for example, it is often necessary to provide for *preliminary drainage and runoff of water* and to apply methods of driving mine openings and stoping which preclude inrushes of water-saturated ground. There is a great abundance of water in the deposits linked with *karsts* and occurring in depressions (for example, bauxite deposits on the eastern slopes of the North Urals). Also abounding in water are the occurrences containing hard but badly fractured rocks lying in lowlands. One example is the Beryozovsky gold ore deposit in the Urals.

The presence of *firedamp* in ore deposits is a rare exception. There are some known cases of local methane accumulations in placers worked by the underground method and other instances of this gas penetrating into underground ore workings from nearby coal seams.

The drilling of holes may cause fine *dust* to accumulate at working places, and inhaling it may entail an occupational disease called *pneumoconiosis*. The most dangerous form of this disease is *silicosis*, caused by silica dust. In addition to the usual measures against this very grave menace (see manuals on industrial hygiene for miners), there is one of considerable importance in coping with the silicosis hazard and that is the choice of the mode of ore breaking. Blasting by long holes is in this respect better than the shallow-hole method of mining.

Consumption of *mine timber* is by far not the same for all methods of mining. Therefore, in considering the modes of wall-rock control, one should take into account the total cost of mine timber, including its delivery to the pit. This problem acquires special importance when one has to choose a mining method for deposits occurring in mountainous regions, where the delivery of timber is particularly dufficult and costly, and where, consequently, preference should be given to the methods involving the least possible consumption of timber.

## 9. Effects of Mechanisation on the Choice of Mining Method

As in coal mining (see Chapter IX, Section 15), the choice of the mining method and mode of stoping for working ore bodies requires that particular attention be given to the selection of machines and equipment necessary to ensure the all-round mechanisation of extraction operations.

The considerable hardness of ores and enclosing rocks frequently encountered in working ore deposits makes drilling of holes and blast-holes particularly important. Today holes are everywhere drilled with the aid of pneumatic jacks or air-legs, which facilitate the work of drillers and enhance the efficiency of air-hammers. The holes are flushed with water (wet drilling). Conditions in the stopes permitting, the drilling machines are set up on jumboes. The jumboes used in the United States are railless, and have automobile wheels or caterpillars.

To raise the efficiency of air-drilling, it is important to increase the pressure of compressed air (up to 7 atm in the stope), this demanding its increase at the compressor plant and reduction to the minimum of pressure losses in the supply lines leading to the stopes. This gave birth to a tendency to set up compressors underground and to arrange special "hydropneumo-storage batteries" to maintain uniform air pressure. At one of the mines in the Urals there is an installation like that with a capacity of about 500 cu m, while the Roedsand iron ore mine in Norway has a hydropneumo-storage battery with a capacity of 2,000 cu m. For underground drilling of long holes with a diameter of 60-130 mm Soviet industry manufactures special rotary drilling machines.

Ore is hauled from the stopes largely by scrapers or slushers provided with hoists powered by electric or air motors; low-power motors (about 10 kw) are in many instances replaced by ones of 20, 45 and more kw. Thus, in the West-German iron ore industry there are scraper hoists driven by 180-kw motors. For stopes with low output and short haulage distances small scrapers driven by motors of 4-5 kw are quite satisfactory. Special "tugger" or service hoists are used for hauling mine timber.

Low-capacity mine cars are still widely used in the ore industry. In the Krivoi Rog mines, rocker-type self-tipping mine cars with a capacity of 1 cu m and carrying power of 2.5 tons are used almost universally. With the size of the mines growing and the introduction of highly effective systems of mining, there is a tendency to adopt large-capacity doorless or self-dumping cars, with a capacity of 2 and 4 cu m and carrying power of 5 and 10 tons.

In the United States there are mine cars with a load-carrying capacity of 25 tons used in underground mining. Also used are self-propelled mine cars with electric or diesel engines and devices for automatic discharge.

The ore in the stopes is loaded into mine cars by loading machines, most often of the types similar to those manufactured by the Soviet industry (ПМЛ and УМП models).

When the ore is mined in continuous stopes-walls, it can be transported by flight or drag conveyers.

The type of mechanisation adopted greatly influences the stoping methods. The change-over from shallow holes to long holes, for instance, has decisively influenced the increase of sublevel intervals.

## 10. Significance of Concomitant Exploration During the Exploitation of Ore Deposits

We have seen that the geological structure, matter composition and conditions attending the occurrence of ore bodies are extremely variable. Consequently, preliminary prospecting and detailed exploration effected in deposits with a view to determining the reserves according to the classes of the pay ore and the conditions of their extraction are by far not always sufficient for steady current mining operations. Therefore, these data have to be constantly kept up to date and supplemented by *exploration work* effected parallel with actual mining. This is done by observations in development openings driven in the deposit and by regularly subjecting the ore to chemical and other analyses. If and when necessary, special workings may be made for exploration purposes, but with a view to their ultimate utilisation in actual mining. Another convenient method of current mine exploration is by driving prospect holes from underground workings with core sampling.

Information obtained through exploration can be properly interpreted only when one has a clear idea about the genesis and peculiarities of the geological structure of the deposit. This emphasises the importance of good knowledge of everything pertaining to the geology of the deposit both for selecting proper mining method and for the conduct of current operations.

CHAPTER XX

# MINING OF THIN AND MEDIUM-THICK ORE DEPOSITS

## A. FLAT AND MODERATE DIP

### 1. Pillar Method of Mining Manganese Ores

Manganese ores in the Nikopol district occur in rocks of the Tertiary age. The ore-bearing bed 1-3.5 metres thick lies in general horizontally, though its bottom is somewhat undulating. The depth of occurrence is insignificant—a few scores of metres. The roof of the bed contains green plastic clay, while the bottom has sand, sandy clay and kaolin and occasionally disintegrated granite. The bottom is subject to strong "bulging". The ore-bearing bed is soft, except for occasional partings of a richer hard ore. In the stope the rock of the bed breaks quite easily. These natural conditions, that is, weakness of wall rocks, require, firstly, limitation of the size of mine fields and, secondly, adoption of a mining method involving the minimum number of workings and stoping by individual and narrow strips.

The opening up of a mine field and the layout of its main development openings were described in Chapter III, Section 2.

The mine field is worked by the retreating method, from the boundaries to the shafts. As the enclosing rocks are weak, all the development openings are supported by four-piece sets made of timber 30-40 cm thick.

Stoping is done by cuts (Fig. 312) five metres wide. From the open goaf these are separated by an ore pillar 1.5-2 metres wide. To start a cut, two "room frames" are set up in the extraction drift, then the props of the drift sets are removed five metres away and the face of the cut is established. Its support is shown in Fig. 312, *I*. The stope of the cut is advanced over a distance of 15-20 metres. The broken ore is hauled from the stope in mine cars. The cut is usually serviced by a faceman and two trammers. This team performs all the operations—breaking of ore by pneumatic hammers, loading it into mine cars, timbering of the cut and laying of mine tracks. Mine-car haulage of ore is giving way to belt-conveyer transportation.

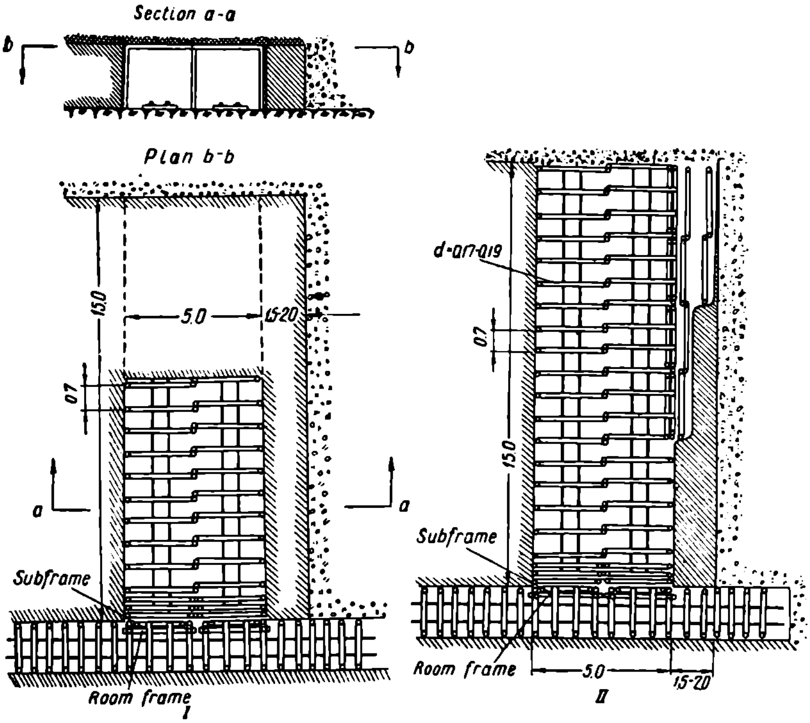



*Fig. 312.* Active stope in mining a manganese ore bed

The ore pillar is robbed by retreating with the method of support shown in Fig. 312, *II*.

After this the timber in the cut is knocked out and the roof is allowed to cave in. When this work is done, it is the turn of the next cut to be extracted in the same manner.

The ore mined in each cut per shift comes to 35-60 tons, with the output per worker engaged in the stope amounting to 12-18 tons. Mine timber consumption is about 50-55 cu m per 1,000 tons of the ore mined.

The extraction of the ore bed by cuts has many disadvantages, which are common to this method in general. Therefore, experiments have been going on in the past two decades with long pillar mining with continuous faces—*walls* 25-30 metres long. Roof control in longwalls has proved a difficult task on account of the properties of the cover rock, and the tests of the continuous face method have so far not led to the substitution of this more progressive method for that of extraction by cuts. According to Y. Shishov, longwall mining methods can also be widely used in the Nikopol manganese district, provided there is an adequate organisation of mining operations—uniform rate of face advance, proper stope support and regular roof control by its caving along the breaker or rib lines.

On the eve of the Great Patriotic War underground *hydraulicking* tests were carried out in this same district. As shown by Fig. 313, the stoping involved cutting the bed into 20-metre-wide pillars and loosening and washing the ore bed by powerful jets of water emitted by hydraulic giants (monitors) at a head of 22-24 atm. The stope remained unsupported and the roof rocks caved in as the strips were extracted. As is usual for hydraulicking, the broken ore was transported along pipelines in the form of pulp. This mode of mining proved highly efficient. But the method had also a specific drawback—"underwashing" of the ore at the bottom of the bed on account of the horizontal occurrence of the deposit and considerable waste of valuable oversize ore, as well as the loss of the mineral in protective pillars (not shown in Fig. 313).

The difficulty of underground mining in the Nikopol manganese district engendered by the weakness of the enclosing rocks, strong

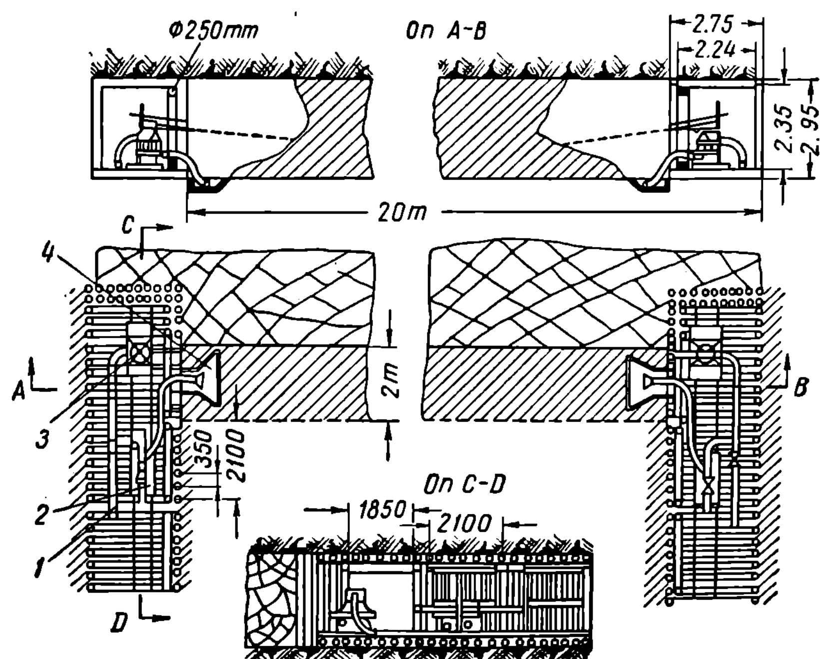



*Fig. 313.* Manganese ore hydraulicking
1—water conduit; 2—hydraulic elevator; 3—hydraulic giant; 4—pulp collector

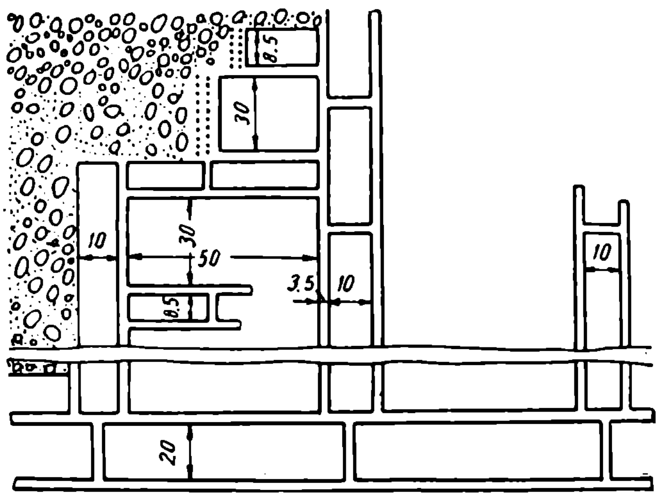



*Fig. 314.* Pillar mining of manganese ore

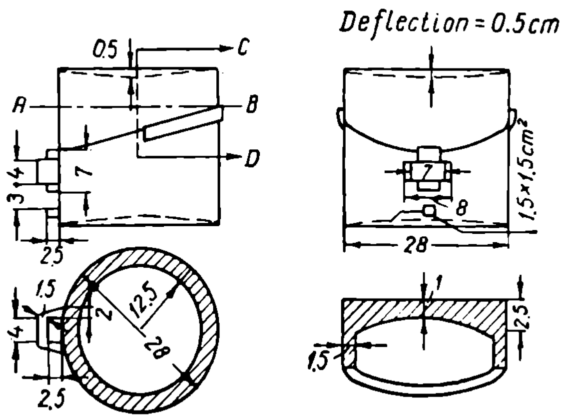



*Fig. 315.* A rest to facilitate knocking down timber props

rock pressure and shallow occurrence of the ore bed is an impelling reason for urging open-cast methods there.

In another major manganese ore district, deposits near the town of Chiaturi (Georgian S.S.R.), the ore bed is 1-4 metres thick and has a very flat dip. The enclosing rocks are of medium stability and, in places, aquiferous. The modifications of the pillar mining method employed, and the room-and-pillar method of working, depend on local conditions. In mining by pillars, the latter have to be about 30-35 (and up to 50) metres wide. One of the varieties of pillar mining is shown in Fig. 314. In the faces of both development and production workings the ore is broken by blasting, the holes being drilled with mounted electric drills (augers). The ore in stopes is hauled by slushers. Roof control is effected by induced caving. To facilitate the pulling out and preservation of mine timber, special *rests* (Fig. 315) made in some mines of the district are placed under timber props, as suggested by G. Tsulukidze. These rests represent round cylindrical steel boxes cut obliquely into two parts. In withdrawing a prop, it suffices to knock out the wedge holding together the two parts of the rest to make the upper part slide down the lower one. This releases the prop from the pressure exercised by the roof and it can be removed from the mined-out area.

## 2. Pillar Mining of Phosphorites

Mined near the railway station of Shchigry, Kursk Region, is a *phosphorite* deposit belonging to the Upper Cretaceous age and distinguished by the following features of geological structure. The phosphorite bed, occurring horizontally at a depth of 20-30 metres from the surface, consists of two benches (Fig. 316): upper *1*—a strongly cemented sandstone phosphorite plate 0.15-0.27 metre thick and lower *2* made of a green sand layer with phosphorite inclusions measuring 4-100 mm. The lower bench is 0.22-0.4 metre thick. The aggregate thickness of the phosphorite beds is 0.4-0.6 metre. The back of the bed is soft, fissured sandy chalk *3*. The bottom of the phosphorite layer contains a strata of green sand *4*, and is somewhat undulating in nature. The ground is fairly moist, but there is no influx of water. The deposit has been mined by long pillars, which are recovered by continuous faces (longwalls). As the bed is of insignificant thickness, to obtain a sufficiently high room near the production face the roof of the phosphorite layer has to be somewhat slashed. The slabbed ground is used

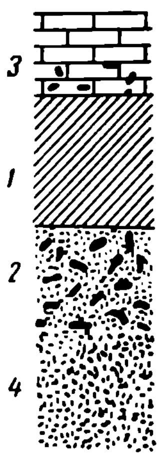



*Fig. 316.* Phosphorite bed structure

for complete filling, and this best solves the roof control problem. The phosphorite plate is much harder than the floor rocks and to facilitate breaking the phosphorite a bottom draw cut is made under the latter.

It should be noted that most of the major phosphorite deposits are shallow-seated, this permitting them to be worked by the open-cut method.

## 3. Placer Mining

Gold-bearing placers as a rule occur at a very flat dip close to the surface and in unstable ground. This favours their surface mining, whose field of effective application is extending progressively thanks to the successes achieved by the Soviet mining machine-building industry. In the underground working of placers it is the pillar methods that are exclusively applied. The opening up of mine fields was discussed in Chapter III.

Since most gold-bearing placers are aquiferous and occur in valleys, the vitally important problem is that of surface water runoff and preliminary dewatering of the placer.

The sequence of *drainage* operations in dewatering a placer depends on the minable placer outline with respect to the stream valley.

The placer hydraulic and drainage operations include:

1) diversion of the river beyond the contours of the pay placer so as to preclude the influx of water into underground workings during the mining process;

2) removal of phreatic water;

3) elimination of any possibility of rain and snow water penetrating into underground workings.

*Diversion of the stream channel* is necessary when the placer lies within an area affected by a river whose water may penetrate into underground workings during the mining operations. To this end a dam is built beyond the minable section of the placer down to the compact water-tight ground (see Fig. 317) and the water is directed along an outfall or water-diversion ditch. The distance between the ditch and the outlines of the pay placer should be 100-200 metres so as to prevent the former from being affected by ground movement caused by the mining of the placer and also prevent the water running along the ditch from infiltrating into underground workings. The possibility of such infiltration depends on the nature of the ground, depth of mining, mining method and local conditions.

The *removal of subsurface water* during the systematic dewatering of a placer is effected with the aid of drainage mine workings made in bedrocks and in the placer itself, the water being subsequently diverted to the surface by a free runoff or pumping method.

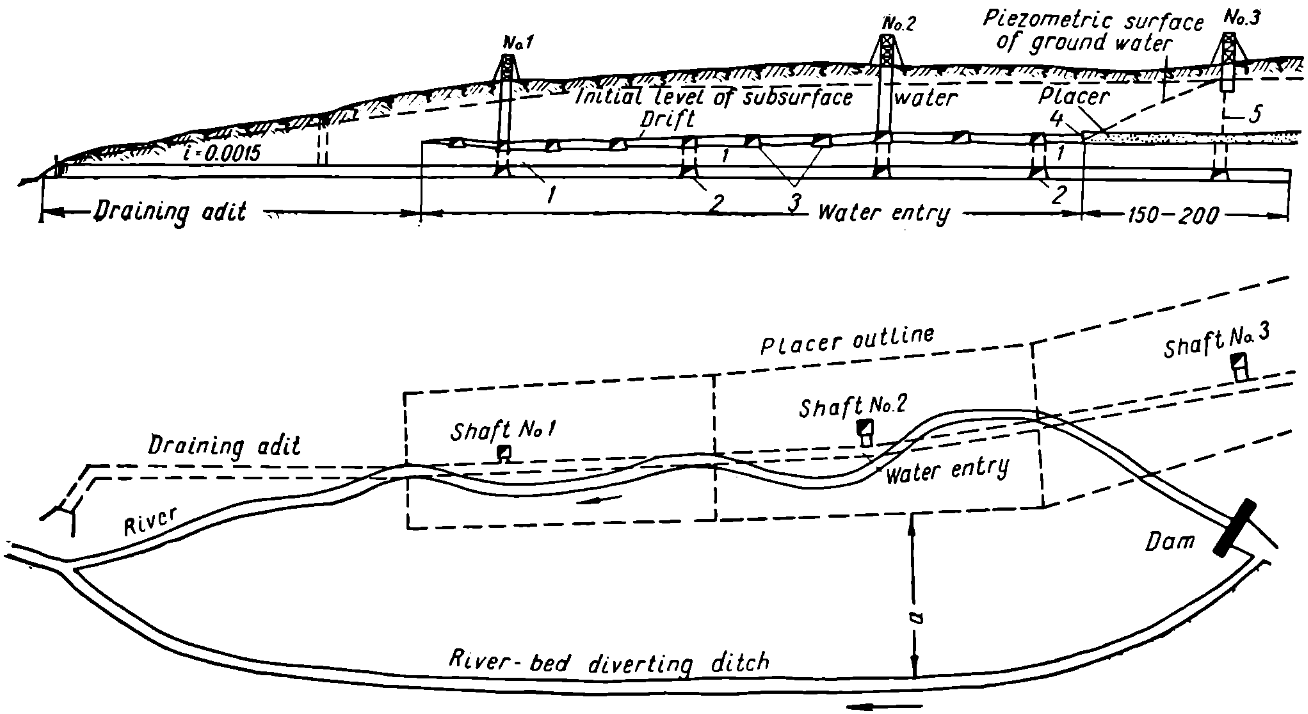



*Fig. 317.* Placer drainage with free water runoff

Fig. 317 depicts the dewatering of a placer by the method of driving an *off-take* drift in bedrocks 2-3 metres below the bottom of the placer, in a zone of rock fracture or in the bedrock strata at a depth of 6-10 metres. In the first instance subsurface water enters this working all along its length through cracks and fractures as it advances. In the second case the off-take drift is made in compact solid ground and gets water from the placer only via through-cuts *1* driven towards the placer from crosscuts *2*. The distance between the through-cuts is 80-200 metres. Advance heading *4* is run in sufficiently drained sands of the placer, 150-200 metres behind the face of the off-take drift, and from this heading crossdrifts *3* are driven towards the lateral boundaries (edges) of the placer.

The time required for the drainage of the placer depends on the intervals between these drifts. The water entering the off-take drift flows to the surface by gravity through a drain adit. To prevent silting, these openings are made with a gradient of 0.0015, and much less frequently—0.001. When the placer occurs at a greater depth, the valley slopes slightly and the water is less abundant, artificial drainage is used and for that purpose a *central* pumping plant is installed in a shaft sunk some distance down the valley.

In both cases offset "technical" hole *5* from which the water flows to the off-take drift is bored during the sinking of the shaft to drain the stope.

The *diversion* or *runoff of rain and snow waters* in placering is effected by excavating *hillside* ditches of a section sufficient to ensure the maximal inflow of water during the spring and storm floods. When a stream-channel diversion ditch runs along one of the valley slopes (Fig. 317), the hillside ditch is excavated only on the opposite slope. When large inflows of water are expected, wooden troughs (*staves*) are laid along one of the slopes.

As stated before, placers are worked by *pillar* mining methods. The pillar-and-bord system has by now almost lost its significance, and it is various modifications of long pillar mining with the recovery of pillars by *continuous faces* (longwalls) or, in the more difficult natural conditions, by slab drifts (strips) that are now employed. In the main, this mining method involves *caving*, and less frequently partial filling.

The development of a mine field begins with the driving of the main drift (Fig. 318) along the thalweg of the placer. When it is driven, this drift is called advance heading. When highly aquiferous placers are worked, an off-take (draining) drift is simultaneously excavated in the bedrocks. In placers of considerable width, a service (auxiliary) drift is made parallel to the main drift, 10-12 metres from it. It is designed for the passage of men, timber delivery, etc. The main

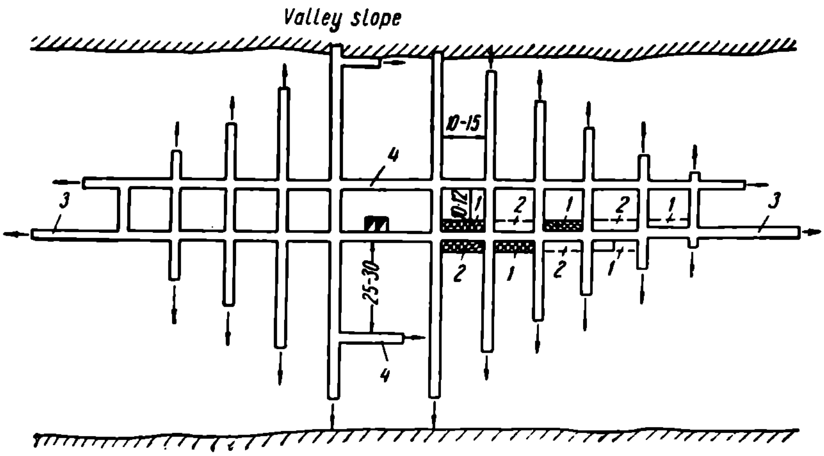



*Fig. 318.* Development of a mine field in working a placer by long pillars
*1, 2*—"waste packs"; *3*—main drift; *4*—extraction drifts

drift has strong timbering (Fig. 319); in unstable rocks even timber sheet piling is used (Fig. 320).

*Extraction* (cross) drifts are run every 10-20 metres from the main drift to the boundaries (edges) of a placer. To protect the main drift from rock pressure and reduce the losses of metal, waste pillars are built on the sides of this opening, with spacers (interlayers) of timber and brushwood (Fig. 321).

Pillars are recovered (Fig. 322) by *slab entries* or *slabbing cuts* (strips) *1*, *2*, *3*, etc., each 3.3 metres wide. As a rule, in one pillar under simultaneous mining there are three slabbing cuts: in one—*gold-bearing sands* are extracted, the second is in the stage of being "cleaned up", and the third is either being prepared for caving or is being caved.

The clean-up of the placer bottom is a specific job in the mining of gold-bearing sands. Due to the geological structure of placers, gold particles quite frequently accumulate in large amounts in the bedrocks underlying the placer and penetrate into hollows and cracks in the bottom. The clean-up is done with picks, shovels and metal brushes. In favourable conditions, the number of slabbing cuts extracted at one time is higher than usual and pillaring is sometimes done on two sides, from the neighbouring extraction drifts.

The sands in the stope of a slabbing cut are either drawn by hand or blasted out. The sequence of extraction in weak ground is illustrated by Fig. 323. More efficient extraction from bottom up (Fig. 324) is possible with a good back of the bed and proper drainage, and provided there are no large boulders in the placer. The slabbing cuts are

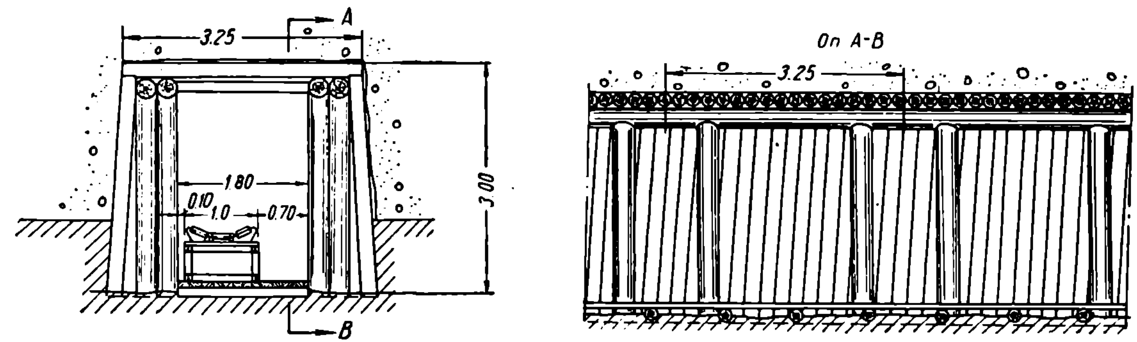



*Fig. 319.* Main drift timbering

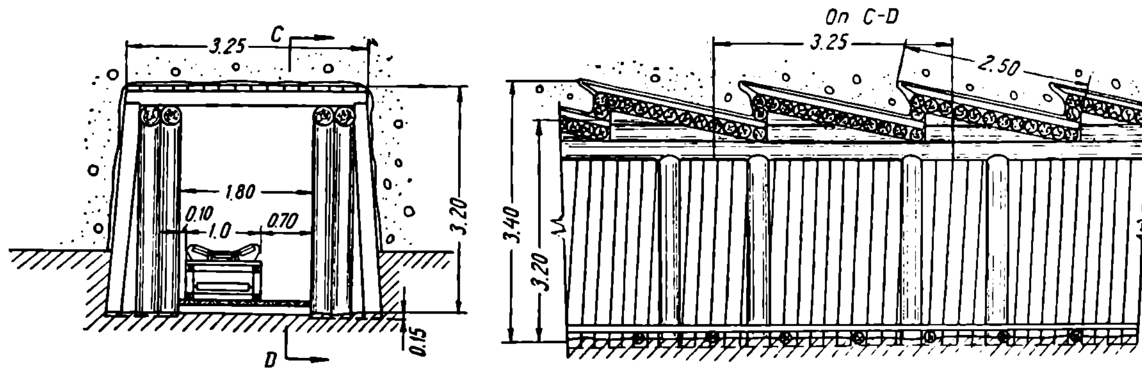



*Fig. 320.* Timber sheet piling

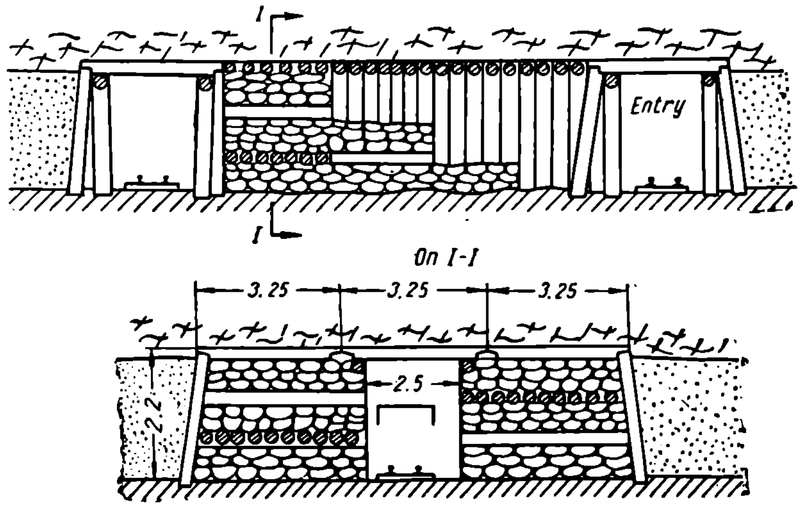



*Fig. 321.* Protection of the main drift by "waste-packs" (pillars)

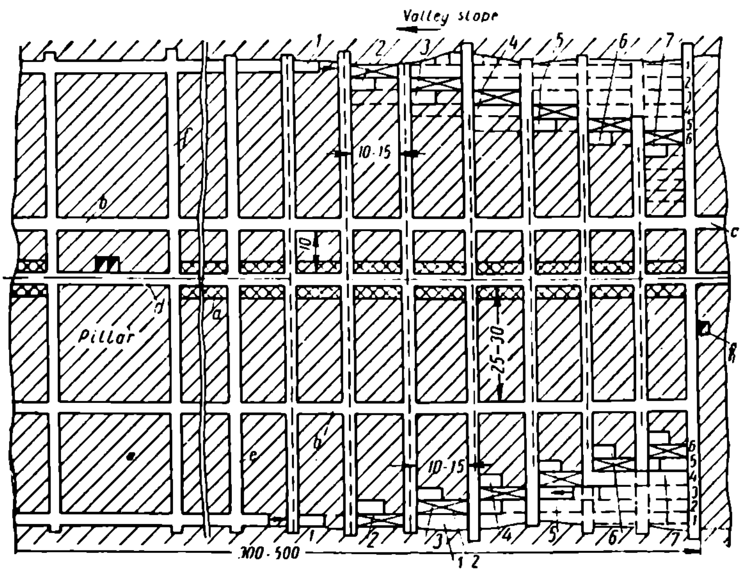



*Fig. 322.* Sequence of stoping operations in placer mining

*a*—"waste packs"; *b*—service drifts; *c*—main haulage drift; *d*—conveyer; *e*—extraction drift; *f*—service shaft

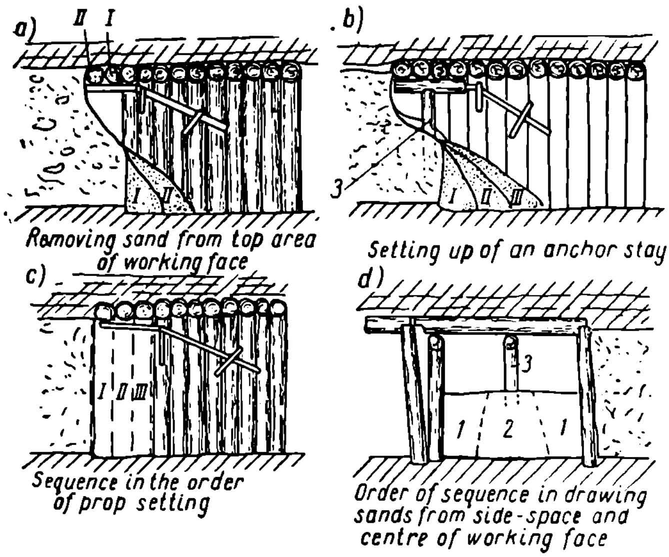



*Fig. 323.* Sequence of sand extraction in the slab entry (cut) of a placer in weak ground

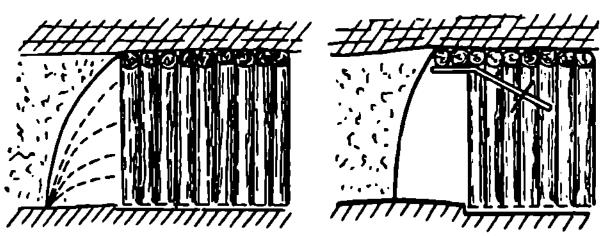


*Fig. 324.* Drawing of sand from bottom up

supported by strong timber sets, similar to those used in drifts (Fig. 325). Hand tramming is being progressively superseded by conveyer delivery.

Roof caving in mined-out slabbing cuts is necessary not only to withdraw some of the mine timber but also to reduce the rock pressure bearing down on adjacent cuts in the stage of stoping. As said above, the caving of a slabbing cut is preceded by a thorough clean-up of the bottom. Before the roof is allowed to fall, the pieces of mine timber are knocked down and extracted by special hooks. The props in the cut facing the stope are not knocked down to obtain a break or rib

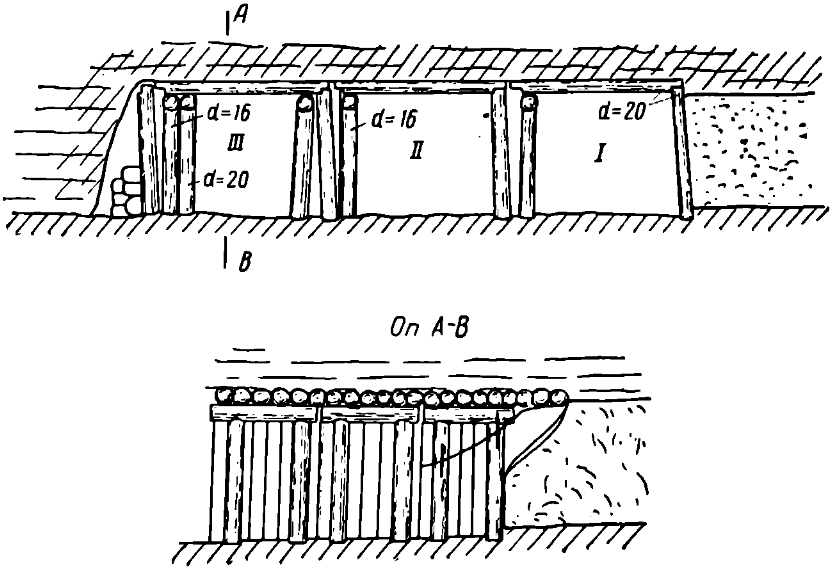



*Fig. 325.* Slab entries support

line for the caving roof. When the crossings are caved, timber is removed by blasting, for these sites are very much squeezed together. In the Lena gold fields 60-80 per cent of mine timber is recovered, with 30-60 per cent of it suitable to be used again. Total timber consumption per 1,000 cu m of sands is in the region of 100-140 cu m.

If the back of the placer has not been drained sufficiently, caving of slab entries is not allowed because there is then a danger of silt-ground inrushes. Pillaring with partial filling of the goaf, involving the building of waste pack walls and cribs, may be employed in this case.

If, not so long ago, underground mining of placers was done almost exclusively by the method of long pillars (and before that by the pillar-and-stall method), in the past few years it is the system of *continuous faces* (*longwalls*) up to 40 metres in length (Fig. 326) that is being increasingly applied in extracting long pillars. *Mechanised* stoping makes it possible greatly to raise the productivity of labour in long-face mining.

Conditions favouring long-face mining are: thickness of cover rocks up to the surface of not less than 10 metres; permafrost ground; dry, stable roof with thawed ground; even bottom and insignificant proportion of boulders in the placer.

The mine field is worked in retreating order. The position of the development openings is shown in Fig. 326. Stoping proceeds in four long faces simultaneously.

The metals and ores in the placer are usually so distributed that they make breast or continuous mining possible. If, in exceptional cases, the thickness of a placer exceeds 3-4 metres, it can be extracted by *slices*, starting with the bottom where the placer is usually richer in valuable components. When the bottom slice has been extracted and fully filled, one proceeds with development and stoping in the overlying slice.

In permafrost conditions, the sands are blasted or thawed. The most popular method is by *blasting* with subsequent thawing of sands on the surface. This method of working frozen sands is simpler and more effective.

The holes should not be less than 1.6 metres long. The height of an active stope or face equals the thickness of the minable portion of a placer, but should not be less than 1.4 metres.

The cycle or round of stoping operations includes drilling, blasting, drawing (mucking) and transportation of sands, timbering and roof control. Stoping is done on the basis of one cycle per 24 hours.

*Thawing of sands* is effected with steam by means of steam-points or pipes. The old methods of thawing by *open fire* have been almost completely ousted by more perfect and effective ones. One of the modes of sand thawing with good prospects is that of underground hydraulicking.

A steam-point is a hollow round steel piece with a diameter of about 25 mm. It is tapered to make it easier to ram into the ground. The steam-points used in stoping are 2-2.5 m long and 3 m long in development work.

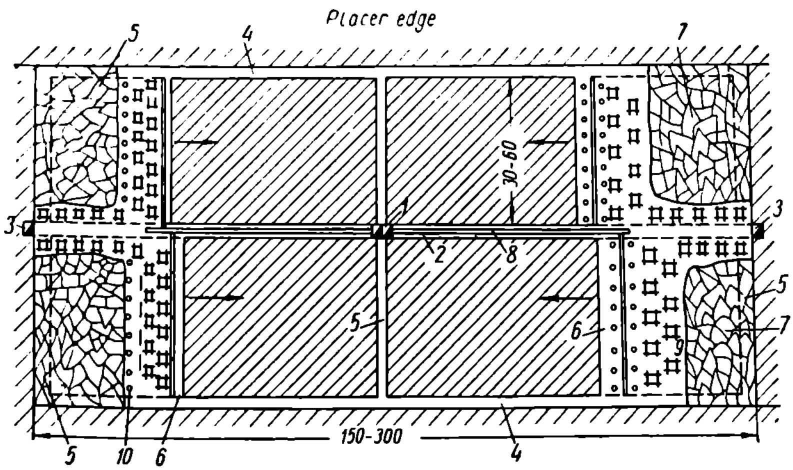



*Fig. 326.* Pillar recovery by continuous faces (longwalls) in a placer

*1*—shaft; *2*—main haulageway; *3*—air pit; *4*—fringe drifts; *5*—extraction drift; *6*—wall face; *7*—caving; *8*—conveyer; *9*—cribs; *10*—breaker timbering

The method of *steam-point* thawing requires preliminarily boring a hole to a depth of 15-20 cm. The point is then driven in and steam is passed through. In the course of the thawing process, the point is rammed into the sand to a depth of 1.5-2 metres (Fig. 327). When the points have been driven in to the full length, they are withdrawn from the holes and replaced by pipes. The mouth of the hole is calked and a maximum amount of steam is let into it. The duration of the individual sand-thawing operations is as follows: driving in of steam-points—2-6 hours; passage of steam via the pipe—from 4 to 12 hours; "sweating" of the ground—10-24 hours. It takes two to four shifts to complete the entire cycle.

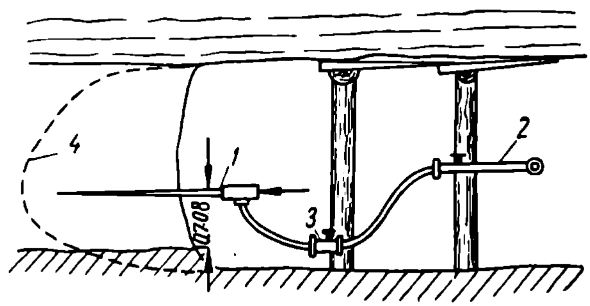



*Fig. 327.* Thawing of frozen sand by steam points
1—steam point; 2—steam pipeline; 3—carriage; 4—boundary line of thawing

When the face is 2.2-2.5 metres in height, the steam-points are placed at 0.75-0.8 metre above the bottom at intervals of 0.6-1.2 metres. If the stope faces are lower, the points are driven in nearer to the floor. The steam is supplied by pipelines from a boiler on the surface. To avoid losses of heat through condensation, the pipes have special thermal insulation. In a more extensive mine field a portable boiler plant moves along with the wall face and steam is delivered there via specially bored holes.

The steam is generated in boilers with a heating surface of 12-36 sq m and more, at a rate of 1-1.5 sq m per one steam-point. The steam pressure in the boiler should be maintained at 6 atm. At the face the steam is distributed among the points through a special carriage and its supply is controlled by special valves.

The advance rate of the wall face per cycle is 1.2-1.7 metres. The output per faceman per shift is in the region of 3 cu m of sand and more. If the placer contains large proportions of clastic material and especially boulders, the efficiency of thawing is liable to drop sharply.

Fig. 328 is illustrative of the pattern involving thawing with *pipes.* At a height of one-third of the stope face (0.6-1 metre) a cut is made 1.7-1.25 metres deep into which a piece of gas pipe 2-4 metres long with an inside diameter of 18-50 mm is inserted. The two ends of the pipe are closed tightly; the surface facing the stope has 2-3 mm holes perforated in it. The pipes are laid along the face at intervals of 0.3-0.5 metre and then covered by a "pile" of sand. To secure the uniformity of thawing, steam is supplied to the centre of each pipe through tubes and rubber hoses. The laying of pipes takes 3-4 hours. "Steaming" takes 6-7 hours and "sweating" 8-9 hours.

The duration of the whole cycle is 17-20 hours. The advance rate of the face per cycle is 0.4-1.2 metres. Output per man per shift is 4 cu m and over.

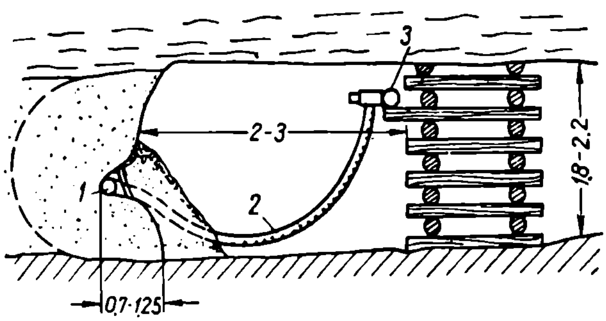



*Fig. 328.* Thawing of frozen ground by pipes
1—pipes; 2—rubber hose; 3—steam pipelines

To the main haulageway the sand is transported by scrapers, conveyers loaded by hand or machine, or by mine cars. Along the main haulageway itself the sand is hauled by mine cars or conveyers. Fig. 329 gives a diagrammatic illustration of scraper haulage along the wall to the haulage drift. A scraper ramp is set up for loading sand into mine cars. Slusher haulage is employed in faces with an even floor and not too great length (20-30 metres).

In walls of considerable extent and productivity the sand is hauled to the main drift by a machine-loaded conveyer (Fig. 330). Moving along the face, the loading machine puts the sand onto a conveyer, which feeds it into mine cars or to a conveyer running along the haulageway. The capacity of the loading machine is 12-25 cu m per hour. The small overall size of the machine makes it possible to use it even in stopes of only one metre in height. The face conveyers most widely used at present are those of the belt and drag types.

The sand brought out of the mine is stock-piled until the seasonal washing.

When sands are mined by blasting, the roof is supported by provisional pillars measuring 5×3, 4×5 and 4×6 metres (Fig. 331), spaced at 10-12 metres across the placer and at 10-20 metres along it.

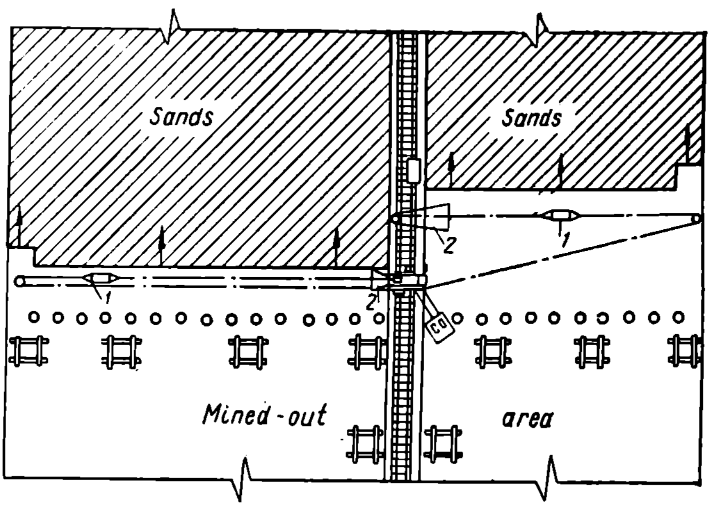



*Fig. 329.* Mechanised mining of sand by longwalls
1—scraper; 2—scraper-loading ramp

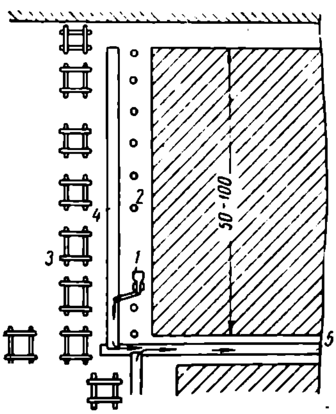



*Fig. 330.* Loading machine in a longwall in placer mining
1—loader; 2—props; 3—cribs; 4—face conveyer; 5—conveyor

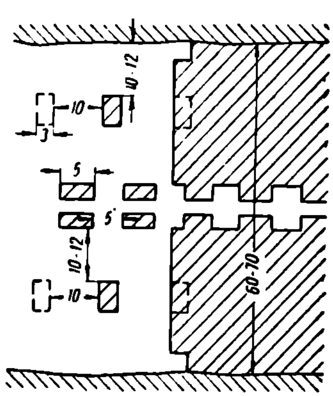



*Fig. 331.* Provisional pillars in permafrost placering

To maintain the haulageway, similar pillars are left every five metres. No pillars are left in narrow placers below 30 metres. As the stopes move forward, the provisional pillars are robbed with the subsequent caving of the roof.

In the thawing of sand by steam, the worked-out space is supported by cribs and temporary face timbering. When the back is sufficiently strong, the timbering includes rows of cribs spaced at 2-3 metres, while runs of props with pads are set up along the face. The roof is broken by removing every third row of cribs and sometimes by preliminarily setting breaker props along the future rib line.

Extraction by longwalls in nonfrozen ground (*taliks*) is not applied widely so far but has fairly good prospects.

Recovery of pillars by the long-face method has a number of advantages: it creates favourable conditions for mechanised stoping; the volume of development work is low—16-18 per cent; the stopes are well ventilated; the cost of mining is low compared to working by slab cuts; consumption of mine timber is reasonable—20-40 cu m per 1,000 cu m of sand; output per faceman is fairly high.

## B. HIGH DIP

### 4. Lode Mining

Extraction of highly dipping low and medium-thick veins is generally effected in overhead stopes, horizontal and inclined slicing and underhand stoping being practised much more seldom.

Preparation for stoping provides for cutting the level by *raises* into working sections, which are termed "blocks" in the mining of ore bodies. Fig. 335, for example, shows a working section extending over 60 metres on strike and 50 metres to the rise, in this instance conforming to the level interval.

Since most of the ore deposits occurring at a steep angle are distinguished by strong ores and firm gangue, the drilling of holes is of exceptional importance in working veins and shoots. Depending on the structure of the ore body, the types of the drilling machines employed and the outline of stopes, the holes may be either horizontal (as in Fig. 337), vertical (Fig. 339) or inclined (Fig. 338). It is more convenient to drill vertical or up-holes with stopers, horizontal and inclined ones with drilling machines mounted on vertical columns.

The systems used in mining steeply dipping lodes differ, this depending largely on the mode of supporting enclosing rocks: the worked-out area may be supported by ordinary or reinforced mine timber, it may be filled with waste, or mining may be done by shrinkage-stoping.

The modes of conveying the ore from the stopes to the lower haulageway depend on the method accepted for supporting country rocks,

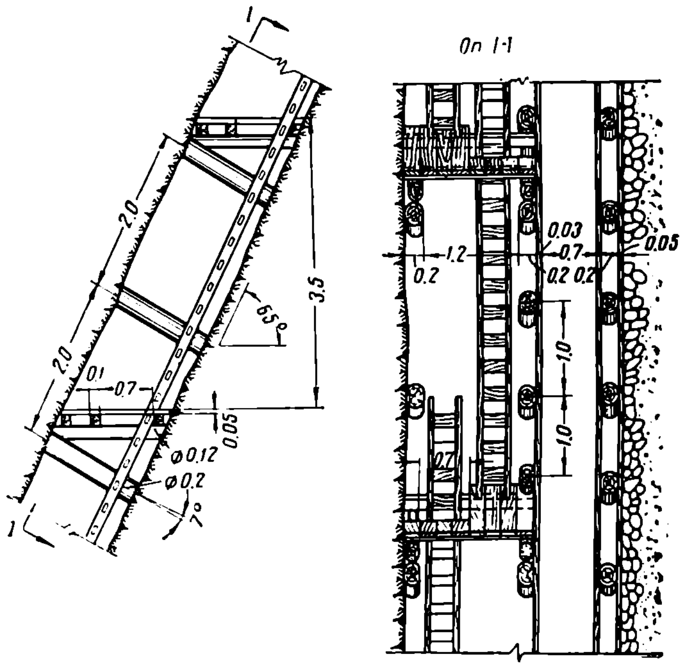



*Fig. 332.* Raises with ladder and ore passage compartments

the disposition of stopes and the wholesale (bulk) or selective mining adopted.

If the mined-out area is supported by stulls, the ore broken in stopes can slide directly down the level. Fig. 332 is illustrative of a raise with compartments for dumping ore and with ladders.

To protect the lower drift, one may employ either ore pillars or only reinforced timber with a round stick flooring (Fig. 334).

Ore (and, if necessary, barren rock too) can be drawn off from the mined-out space through special chutes (Fig. 333).

Below are some typical examples of methods for working low steep veins.

## 5. Stull-Set Method of Mining

Fig. 334 depicts a system applied in working a quartz gold-bearing lode about one metre thick. The position of the overhand stope is clearly shown in the drawing. The ore is very hard, according to Protodyakonov the relative hardness equals 15. The enclosing rocks—diorites—are hard too but capable of forming loose slabs in the back. The stoped-out space is supported by ***stull-sets***, that is, by posts blocked against the hanging and foot walls. Inasmuch as the vein has a steep

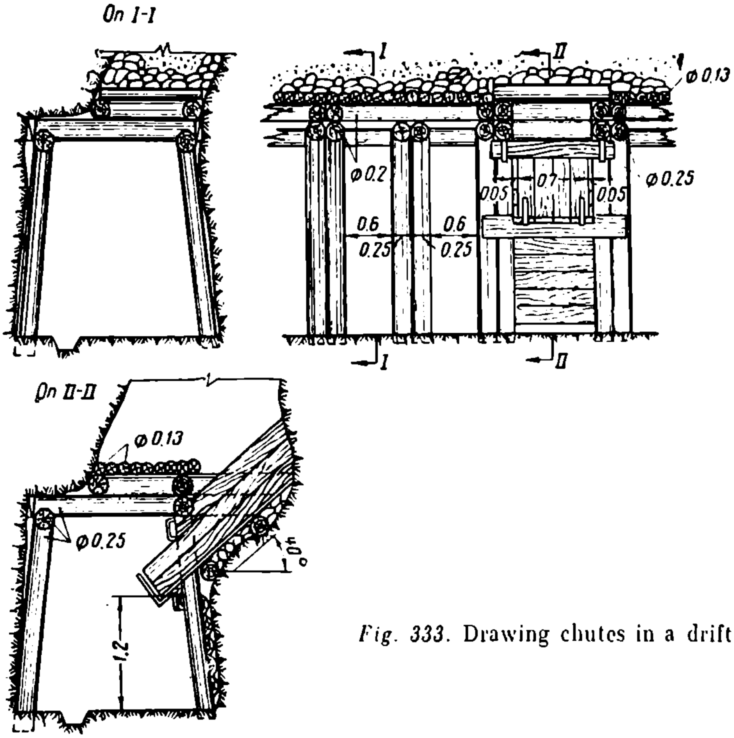



*Fig. 333.* Drawing chutes in a drift

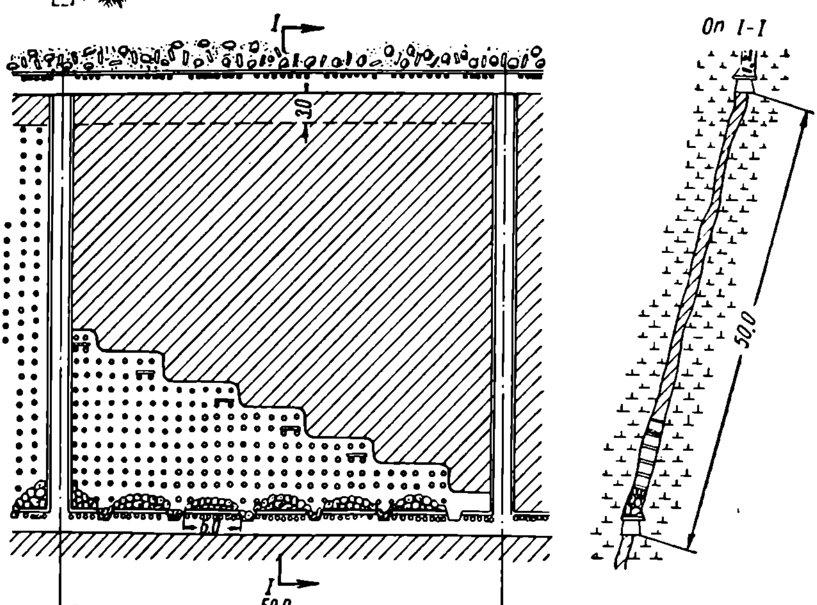



*Fig. 334.* Overhand stoping in a vein with stull-sets

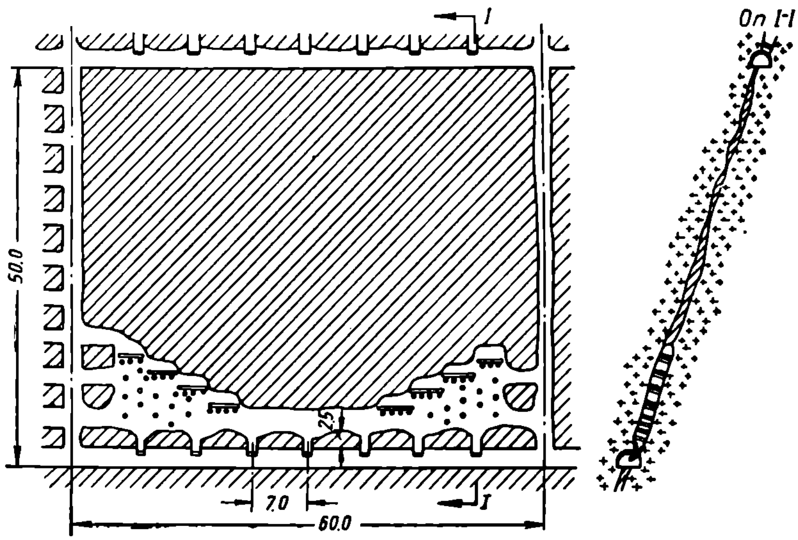



*Fig. 335.* Overhand stoping in a vein from opposite directions

dip of about 50° to 80°, the miners stand with their drilling machines on *laggings* temporarily laid down the posts of the sets. The broken ore slides down into the worked-out area where it is loaded into mine cars through ore chutes. Since the rock is strong and the ore valuable, no floor pillars are left over the haulageway.

The output per faceman per shift is 1.25 cu m, explosive consumption per 1 cu m of ore—1.34 kg, and mine timber consumption per 1,000 cu m of ore—110 cu m.

When ore is strong and wall rocks are firm, the overhand stopes can be carried in the opposite direction so as to enlarge the face front line (Fig. 335).

The stull-set method of mining has substantial disadvantages: the necessity of performing work from a temporary flooring laid over the mined-out area requires a great deal of attention and circumspection on the part of the men engaged in the stopes; setting of timber and its delivery to the stopes are rather difficult; blasting often knocks down old timbering; sticks falling down the level may hamper the drawing of ore from the chutes.

## 6. Mining with Reinforced Support

The support of mined-out space with stull-sets alone can be adequate only in hard and stable rocks of the hanging and foot walls. Stronger support is required if these rocks are liable to cave in. An example

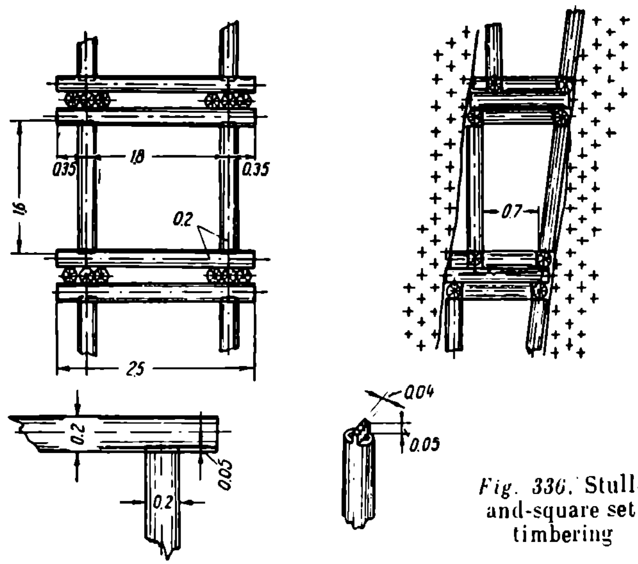



*Fig. 336.* Stull-and-square set timbering

is given in Fig. 336. It shows the mining of a gold-bearing steep quartz vein of an average thickness of 0.35 metre, occurring amidst fissured, altered and unfirm granites. The support in the worked-out area in this case consists of *stringers reinforced with upright props.* Here individual pieces of timbering are made to support each other, thus ensuring the general stability of the set as a whole. This design is used fully in the *square-set* supports described below (Chapter XXI, Section 5), which are sometimes employed in working of thick deposits. The type of support depicted in Fig. 336 is therefore called *stull-square-set* timbering. A drawback of this mode of support is the very appreciable consumption of mine timber, amounting to 220 cu m per 1,000 cu m of ore mined. Ore losses, on the other hand, are low (5 per cent) and, considering the high value of the mineral, this is of prime importance.

### 7. Filling Methods of Mining

Yet another method of maintaining the equilibrium of the rocks enclosing a lode, besides timber support, is the *filling* of stoped-out space. In the working of narrow veins, the filling materials may be the rocks of the hanging and foot walls blasted during the extraction of the vein. Fig. 337 is illustrative of a system applied in working a narrow highly dipping vein (averaging 0.2 metre). Some of the blasted wall rocks are used for filling. The waste pack walls are separated by laggings-off. The ore is lowered to the haulage drift by the

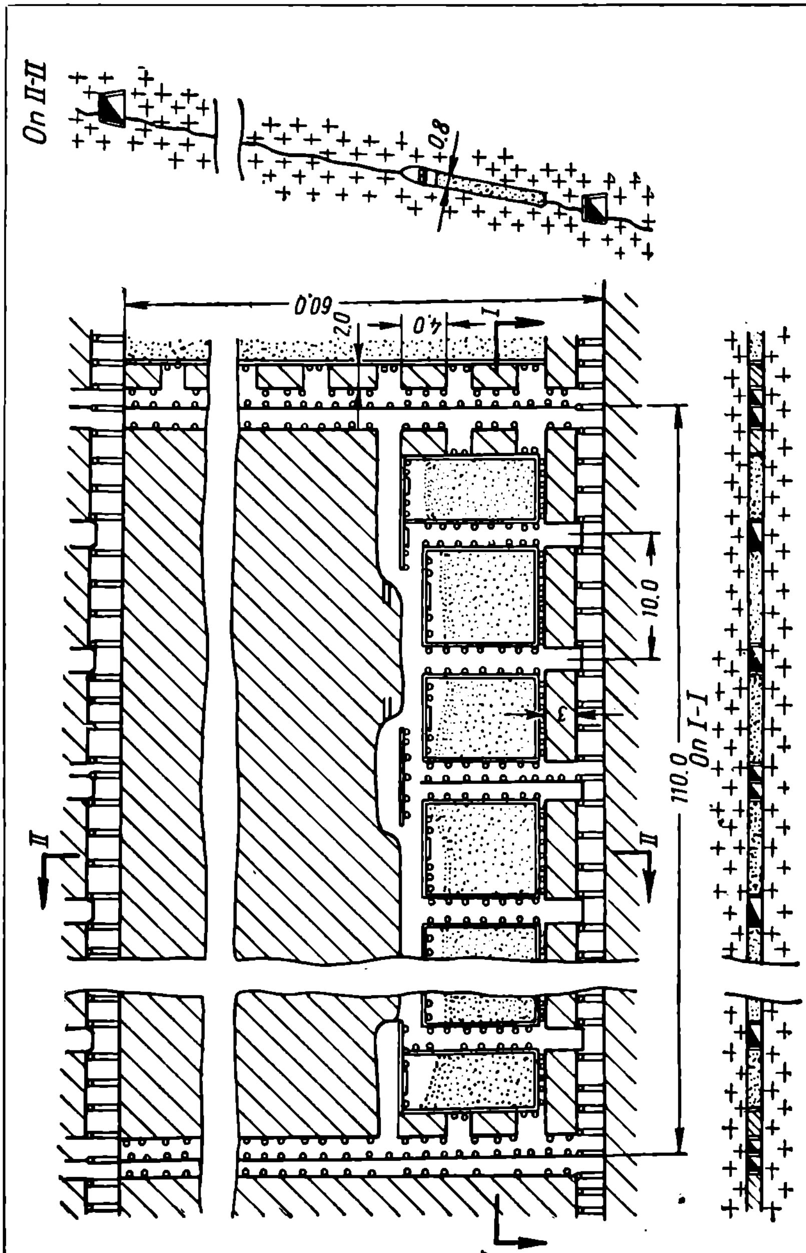



*Fig. 337.* Working a narrow vein with fill obtained in the production stope

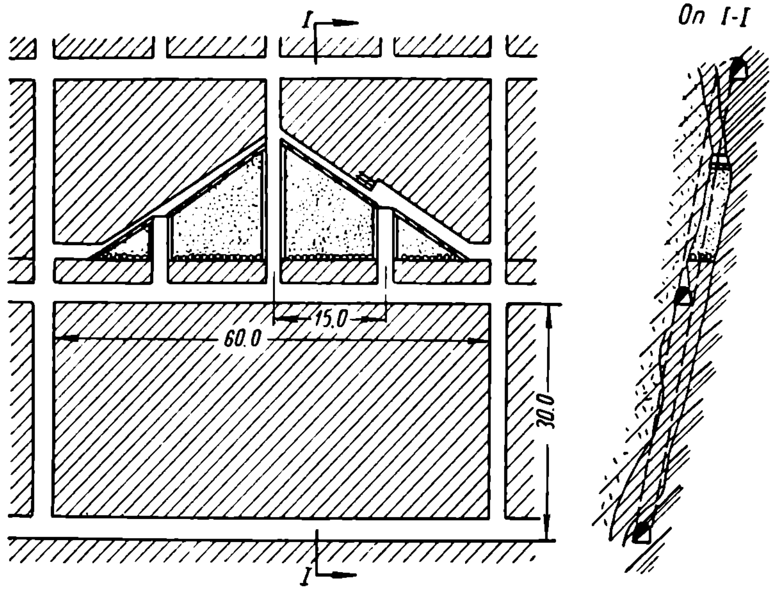



*Fig. 338.* Mining an ore body with fill supplied from outside

*ore chutes* left between the pack walls. These chutes are put up every 10 metres. The ore is extracted in individual benches, each served by its own ore chute. The use of filling appreciably cuts down mine timber consumption and at the same time ensures a low proportion of ore losses (up to 5 per cent).

If the vein is more than 1.5-2 metres thick and the ore body has no gangue inclusions, the use of filling makes it necessary to bring it to the stoping area from outside. An example of this system is shown in Fig. 338, which depicts the mining of a sulphide ore deposit about 2 metres thick, dipping at an angle of 45-80°. On strike the block extends over 60 metres and is 30 metres high, the floor 60 metres high being divided into two sublevels. Work with filling has been adopted in this case because of unfirm wall rocks. A sloping fill pass is driven in the centre of the block to deliver the filling material from the upper drift. The stope is made inclined to facilitate passage of ore by gravity to ore chutes set up near the boundaries of the extraction block, as well as the distribution of the fill in the mined-out area. Consequently, in this case the ore is extracted by *inclined* or *rill slices.* To the haulageway the ore is likewise lowered through ore chutes arranged in the fill itself. The distance between the surface of the mine-fill and the ore stope is such as to leave sufficient space for work in the area near the active stope.

If the amount of waste obtained in blasting wall rocks or gangue inclusions in the ore body is not sufficient for the complete filling of the stoped-out area, it may be stowed *partially*: the waste is then placed on *stulls* covered with lagging, as shown in Fig. 347 (see below).

## 8. Shrinkage-Stoping

In certain conditions, the back in the worked-out area (and in steep dip the bottom as well) can be supported by ore *temporarily kept* in the stope instead of work fill. This ore serves as a temporary fill and at the same time as a floor for the stoping area.

An example of shrinkage-stoping applied in working a narrow (0.7 metre), steeply dipping (40-60°) gold-bearing quartz vein is shown in Fig. 339. The rock contained in the vein is very hard; the hanging and foot walls include firm granites. The ore is broken in an overhead stope and then kept in it. As the ore disintegrates, its volume increases approximately by one-third. Consequently, some of the shrinkage-stoped ore has to be drawn off at the bottom of the level and discharged into mine cars through ore chutes. To ensure that the ore is completely drawn off and eliminate its overhanging, special funnel-shaped hoppers made of timber are arranged over the chutes. Shrinkage-stoping continues all through the process of block extraction. This is followed by the *drawing off* of the stored ore. To ensure proper discharge, the ore body must have a steep regular dip and the enclosing

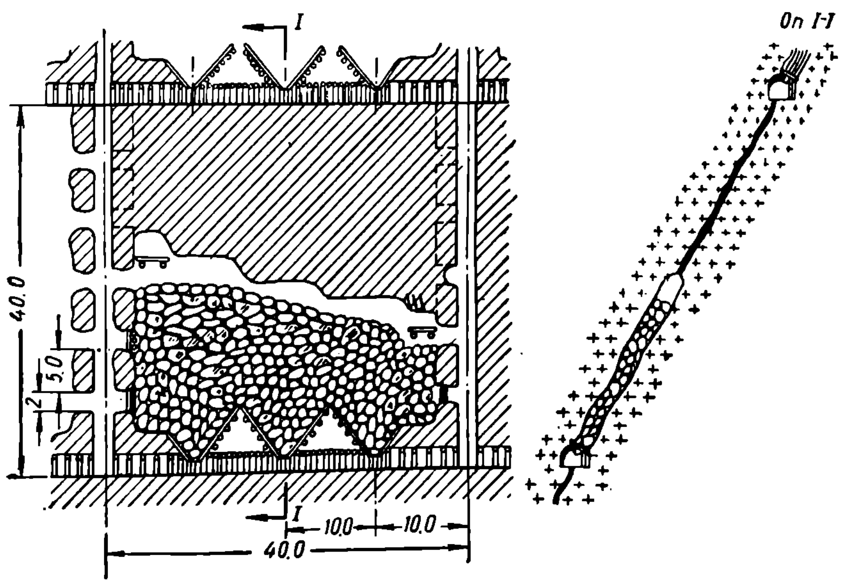



*Fig. 339.* Shrinkage-stoping of a narrow, steep vein

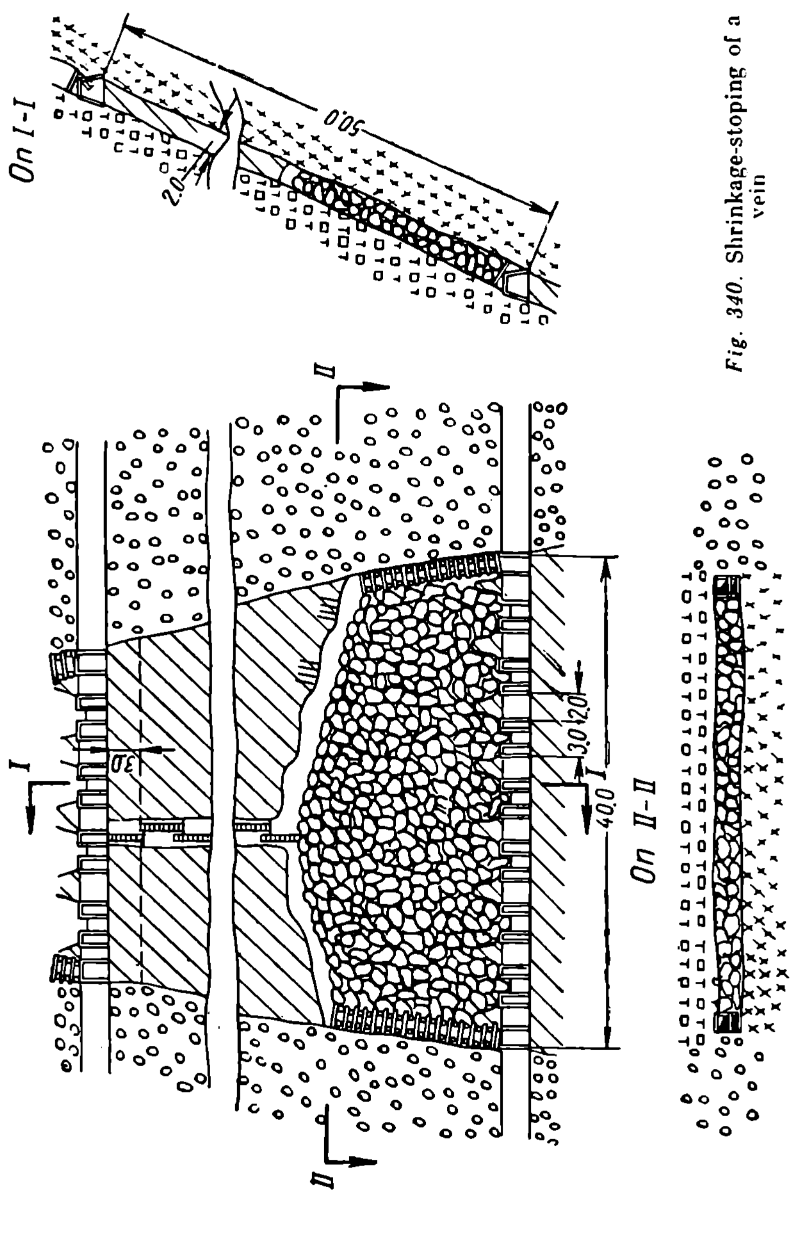



*Fig. 340.* Shrinkage-stoping of a vein

barren rocks must be firm, as is the case in the example under discussion. After the ore has been drawn off, the mined-out space remains without any fill at all. For this reason, to prevent any mass movement of the ground, which might also involve the adjacent extraction blocks in the stage of stoping, the raises delimiting each block on strike are put up with protective pillars on their flanks. The size of the extraction block on strike should be such as to eliminate any possibility of wall rocks caving in within its bounds, at least until the ore has been drawn off completely.

An interesting example of shrinkage-stoping is illustrated by Fig. 340. In this case the size of the extraction block on strike (40 metres) is limited by the natural structure of the deposit. The passageways serving for communication with the stopes are therefore driven with a certain slope along the contacts of the vein with enclosing country rocks. In view of the small size of the block, the central raise serves as a connection with the upper drift.

The ordinary method of ore breaking in a stepped face of the shrinkage-stope becomes hazardous and therefore inapplicable if the ore is liable to cave in because of its physical and mechanical properties. In instances like this, the ore can be broken from preliminarily driven raises. An example of this method is shown in Fig. 341. Here the vein is 1-3 metres wide and dips at an angle of 70°. The ore is weak and liable to cave in spontaneously. Although the enclosing rocks of medium stability do not exclude the application of shrinkage-stoping, the tendency of the ore to cave in makes it necessary to break it from raises put up every 5 metres, as shown in Fig. 339. The driving of raises increases expenditure, but then the drillers are safe. The block is 50 metres long and 42 metres high. The output per faceman per shift reaches 2 cu m; because of the need to support the raises, timber consumption is somewhat high—up to 0.3 cu m per 1 cu m of ore.

Mining conditions needed for successful shrinkage-stoping of narrow veins include: relatively stable enclosing rocks, angle of dip of not less than 55-60°, thickness of the ore body at least 0.7-0.8 metre, regular attitude. Besides, when stored long in a shrinkage-stope, the ore should not be allowed to compact and oxidate.

### 9. Shrinkage-Stoping by Slices

The Institute of Mining of the U.S.S.R. Academy of Sciences (M. Agoshkov, D. Bronnikov, A. Nazarchik and Z. Terpogosov) has proposed, elaborated and introduced in a number of mines an original variation of shrinkage-stoping—so-called *shrinkage-stoping by slices.* The main points of this modification, which, like the preceding ones, is intended for working narrow steep veins, are explained in Fig. 342.

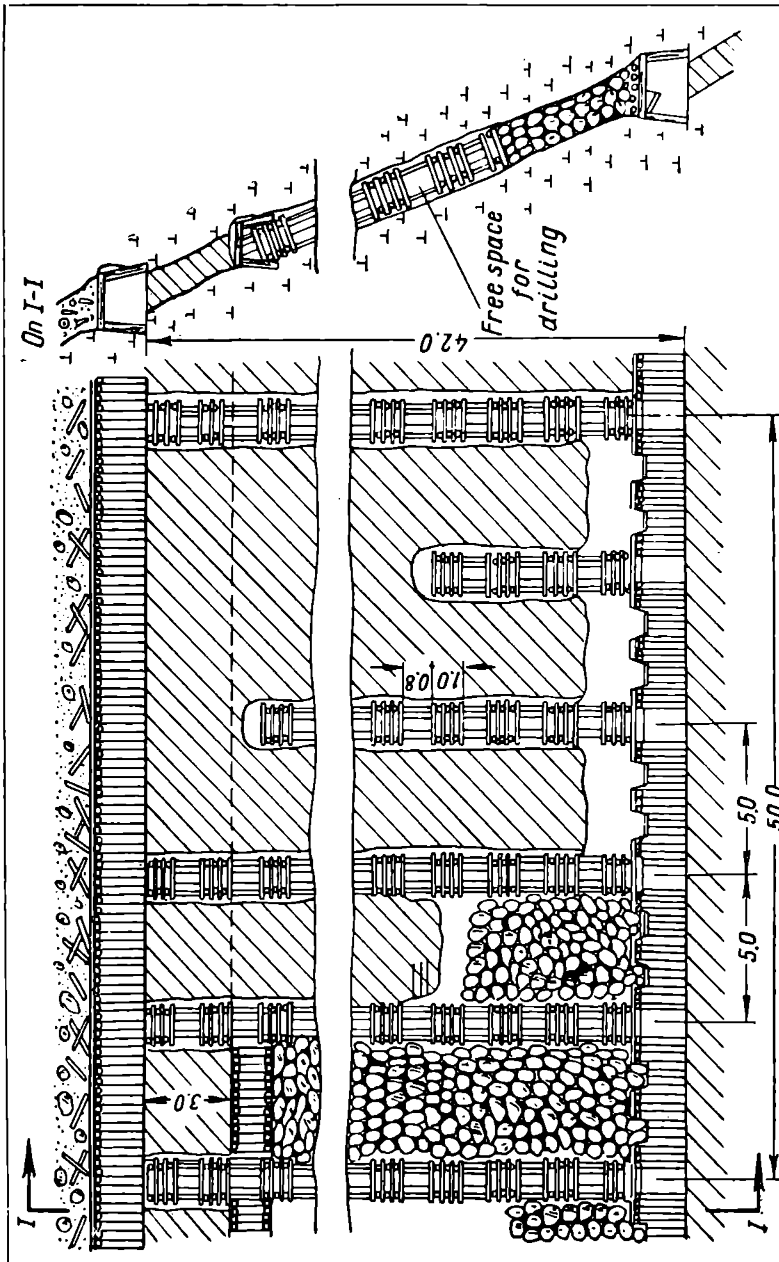



*Fig. 341.* Shrinkage-stoping with the ore broken from raises

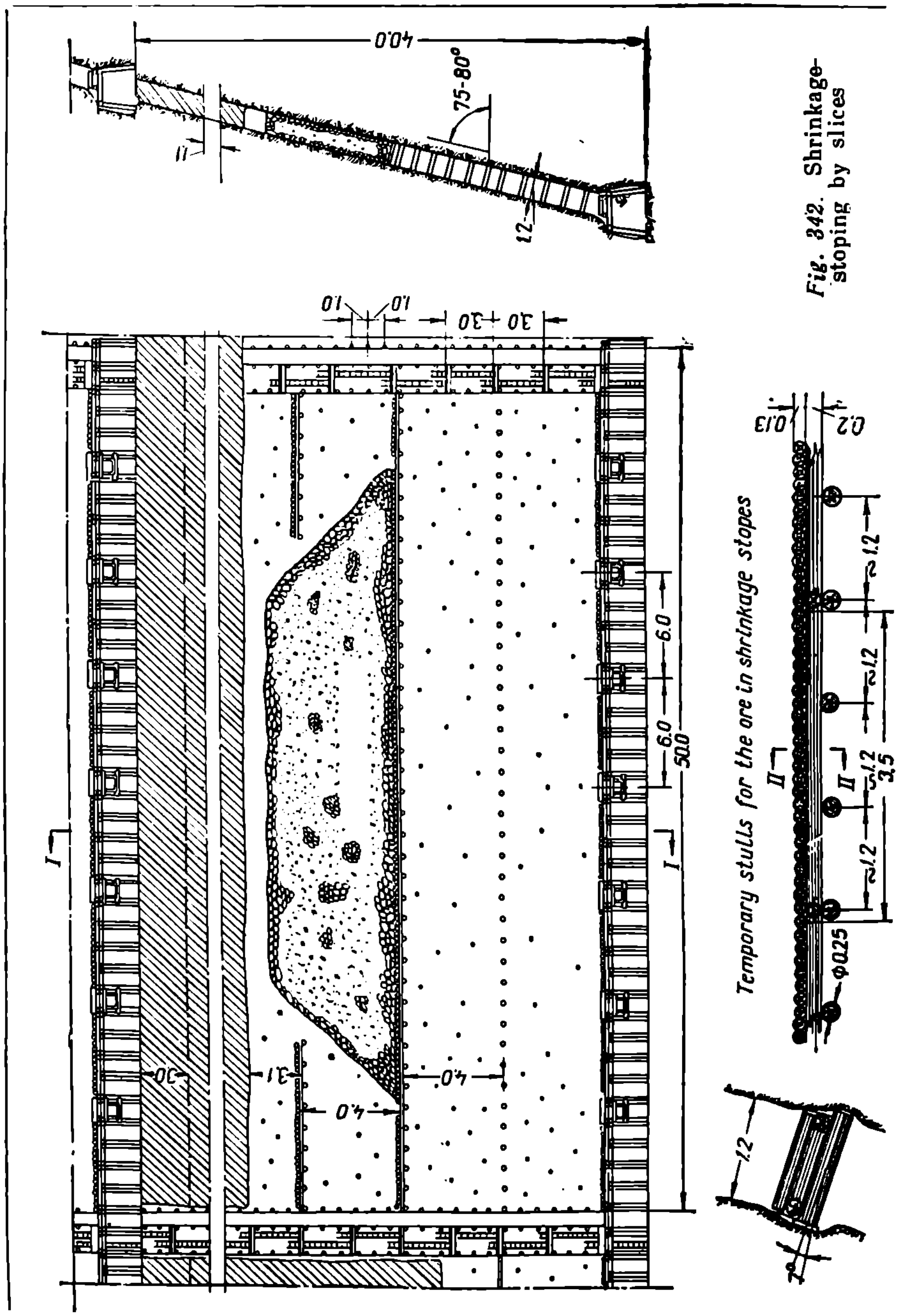



*Fig. 342.* Shrinkage-stoping by slices

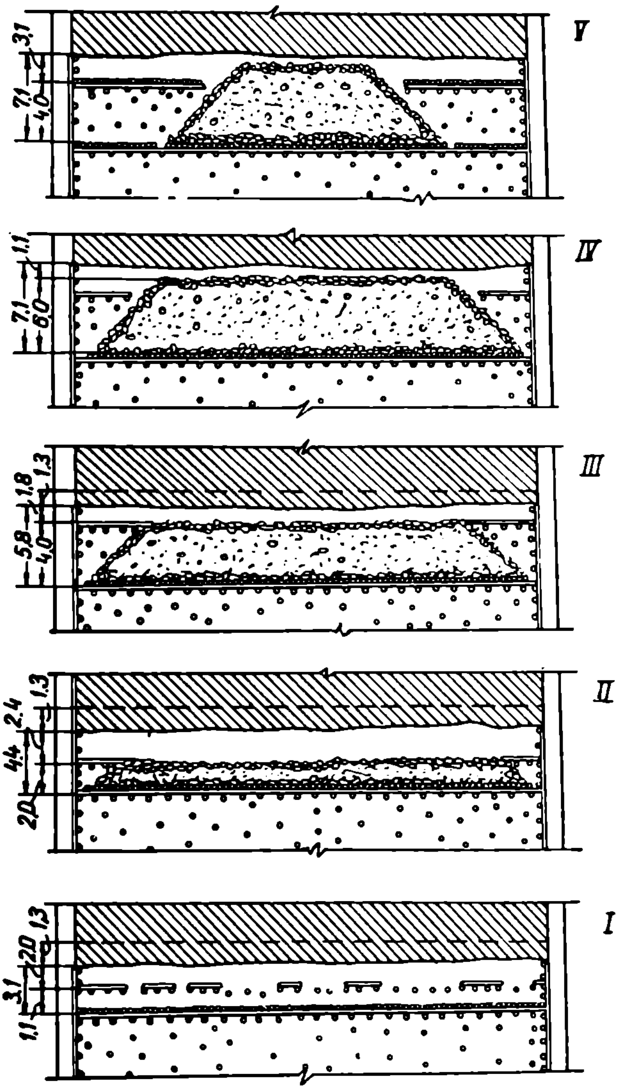



*Fig. 343.* Sequence of breaking, storing and drawing off of the ore

In this modified form of shrinkage-stoping the ore fills only a part of mined-out space. The broken ore is placed on special floorings made of round timber ("dead floor"), resting on purlins. The latter are put on tightly wedged stull-sets. These floors are spaced 4 metres apart. Hence, the massif of shrinkage-stoped ore is shaped as a strip, equal in length to the size of the extraction block on strike, and 4-6 metres high. The ore is broken partly from the stulls and partly from the surface of the stored ore. In the process of drawing the ore is passed down through the flooring which is stripped of some of the lagging boards to make "windows", via which it slides down to the lower drift along the worked-out area. Fig. 343 illustrates the sequence of drilling, shrinkage-stoping and drawing operations: *I*—the beginning of ore breaking with stoper drills from temporary floorings; *II* and *III*—drilling of the stope face from the surface of broken ore; *IV*—shrinkage-stoped ore before its drawing; *V*—drawing of the ore and building of lagged stulls (floorings) for the extraction of the next strip.

This variant of shrinkage-stoping may be employed in mining conditions similar to those in which the stull-set method and ordinary shrinkage-stoping are practised. In contrast to the first of them, the work is done in safer conditions, this contributing to the higher efficiency of the facemen. Moreover, the method makes it possible constantly to have available a certain reserve of broken ore. But this reserve of stored ore should not be so great as in an ordinary shrinkage-stoping, and its drawing off is less complicated.

## 10. Mining with Shrinkage-Stoping of Waste Fill

As stated earlier, the extraction of narrow steeply dipping veins may be wholesome (bulk) or separate (selective).

Bulk extraction of the ore and enclosing barren rocks in very narrow (not more than 0.3-0.4 metre) veins entails considerable dilution of the ore.

Separate mining of mineralised vein and its wall rocks in thin lodes is technically difficult and causes losses of ore. When this method is employed, rich ore has to be broken and passed through discharge chutes separately from gangue. If excessive, the latter is partly used for filling and partly sent to the lower drift via waste dump chutes. To avoid losses of rich ore and its mixing up with the barren rock, special floors of boards, iron sheets, etc., are arranged in the stope. Rich ore fines, however, penetrate into the fill through interstices in the flooring and via chinks in the ore-passes lining.

For these reasons, the Institute of Mining of the U.S.S.R. Academy of Sciences has proposed a novel method of mining providing for *separate extraction of ore and shrinkage-stoping* of the enclosing waste rocks.

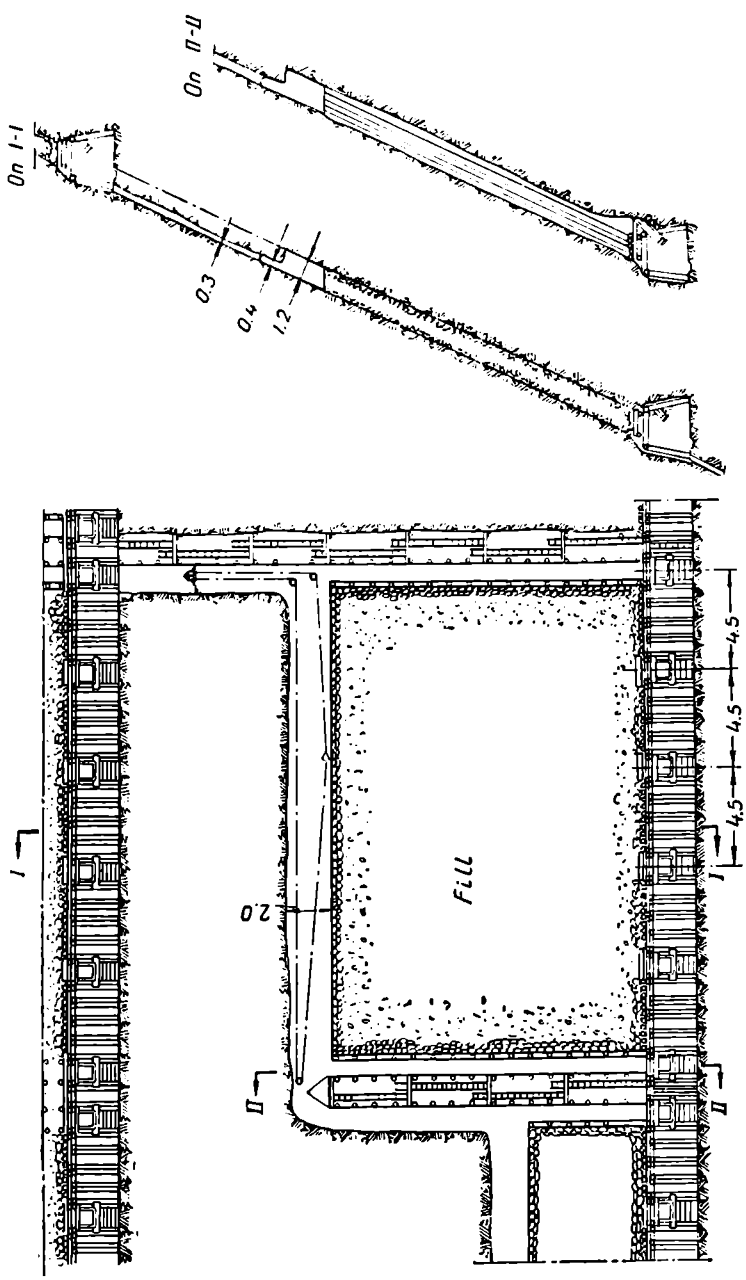



Fig. 344. Shrinkage-stoping of waste fill

In this instance (Fig. 344), the ore is excavated separately and transported to ore passes along a flooring laid on the pile of the fill because the broken gangue is stored in the same fashion as the ore in shrinkage-stoping. In other words, the broken barren rocks enclosing the vein are used to fill the worked-out area, but instead of being passed down via dump chutes, the surplus is discharged at the bottom through waste passes directly into the lower drift. If the enclosing rocks include any ore matter in quantities warranting its profitable extraction, the filling material can be drawn off from the shrinkage-stope on the completion of block excavation. Ore drawing chutes are put up every 3-6 metres.

To break the vein, up-holes 0.8-1.2 metres deep and 0.3-0.6 metre apart are drilled in the stope (Fig. 345). The holes should preferably be of smaller diameter—around 30 mm—and explosives sticks correspondingly smaller. The vein filling is blasted onto a flooring made of steel sheets 8 mm thick, placed end on end. Pieces of an old conveyer belt may also serve as flooring.

The broken ore can be taken out of the stope along ore passes or via steel (with a 0.5-metre bore) or wooden (not shown in Fig. 344) pipes which are gradually extended in the mass of the fill.

When the vein has been extracted, the rocks of the foot (Fig. 344) or hanging walls are broken and are subsequently partially utilised for filling.

The system is distinguished by the following advantages: selective excavation and drawing of rich ore; absence of labour-consuming work in driving a large number of ore chutes; possibility of a large number of drillers working to good advantage in one stope; low losses of ore; insignificant consumption of mine timber.

## 11. Sublevel Back Stoping

A drawback of selective extraction lies in the fact that, notwithstanding separate breaking and drawing of ore, it is necessary to blast and slash the foot or hanging wall of the vein in order to obtain sufficiently large space in the stope for men to work in. Interesting in this connection is the method proposed by engineer V. Mertsalov. It involves breaking of the mineral from *subdrifts* without blasting the enclosing rocks. As shown in Fig. 346, the subdrifts are driven immediately behind the stopes, 4-5 metres apart. This small distance is chosen in order to facilitate breaking the vein matter by drilling up- and down-holes from the subdrifts. The shorter holes are drilled first, the longer ones after. The broken ore slides down to the bottom of the level via the worked-out area. The barren rock blasted in sub-

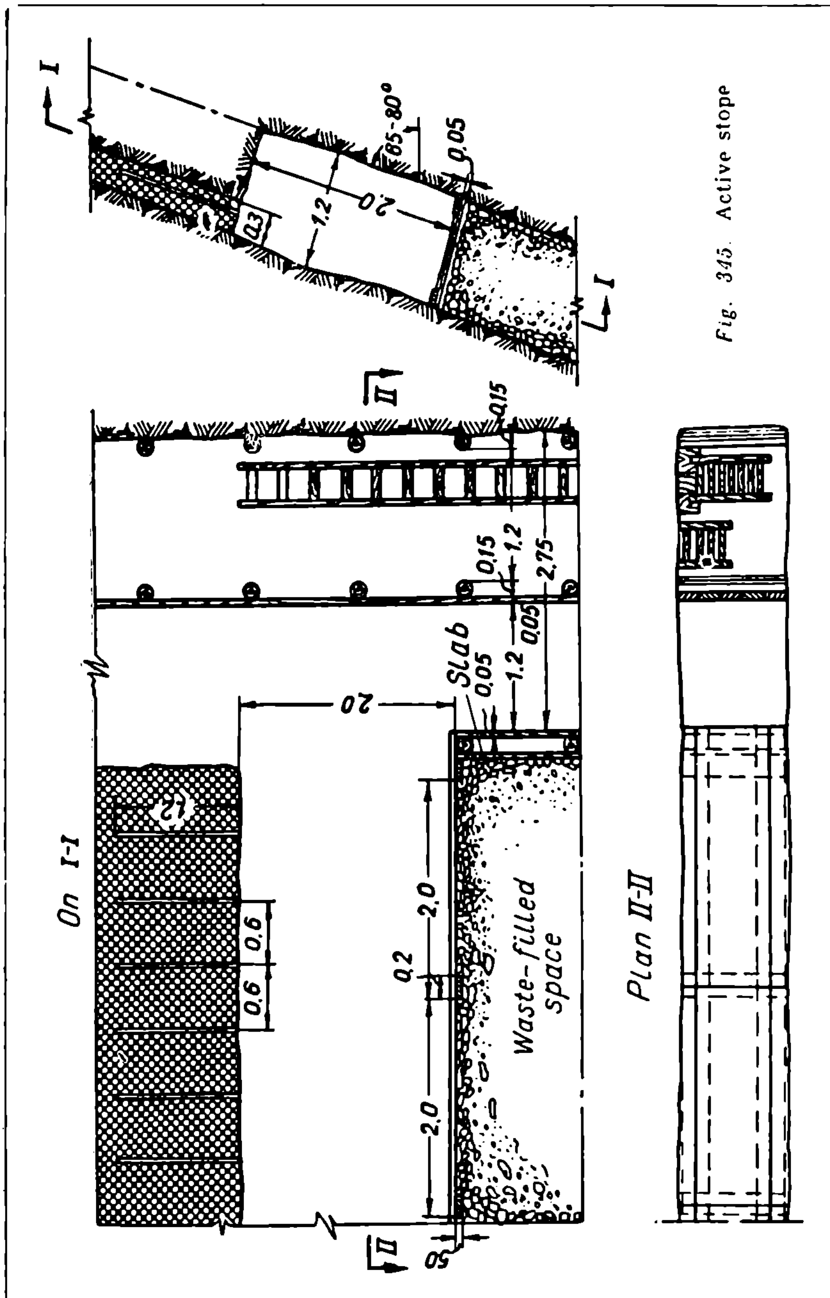



Fig. 345. Active stope

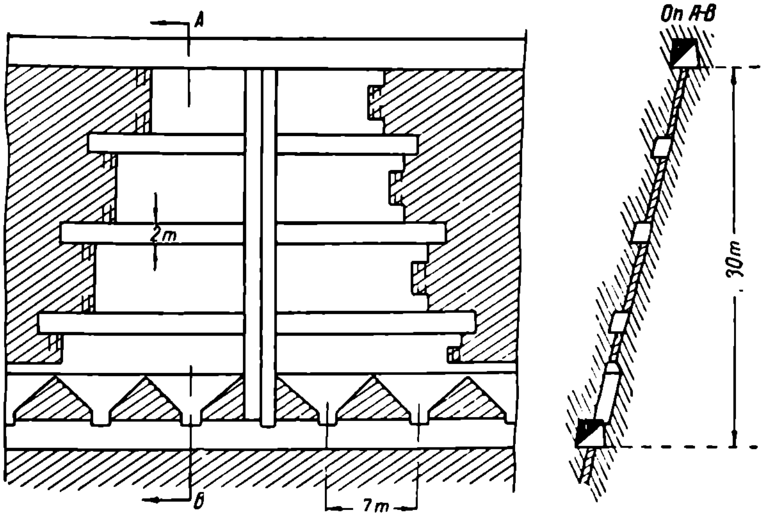



*Fig. 346.* Sublevel drift stoping of a narrow vein

drifts may be lowered to the section of the open goaf no longer used for the dumping of the ore.

This method enhances the efficiency of facemen, widens the working front and keeps the consumption of mine timber down. On the other hand, the application of the system causes greater ore dilution than in ordinary selective mining.

## 12. Basic Notions of Underhand Stoping

While an overhand stope resembles an overturned staircase, the *underhand* one (Fig. 347) may be compared to a staircase in an ordinary position.

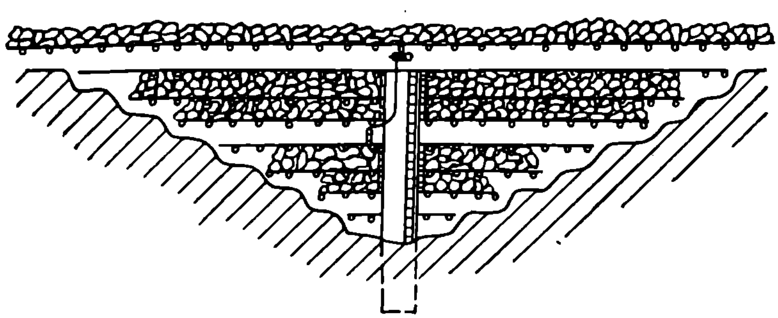


*Fig 347.* An example of underhand stoping

Blasting in an underhand stope is less effective than in an overhead one. The ore in benches has to be reshovelled. The men in the face have mined-out space over their heads and not solid ore, as is the case in overhand stoping.

For this reason, work with underhand stopes is practised very seldom, almost exclusively in extracting small portions of ore bodies which peter out somewhat below the lower haulageway and which can be mined out down the dip more easily, thus eliminating development work on the next floor.

Fig. 347 represents a scheme of this mode of mining. The stopes are opened from a blind shaft. If the work involves filling, the waste is placed on heavy stulls covered with lagging. The latter also serve to protect men working in stopes from falling rocks and other objects. Drifts are left in the pile of fill for the haulage of ore to the hoisting plant. The ore is hoisted in low-capacity skips by an electric or air hoist set up in the upper drift. The blind shaft is also furnished with ladders. In the case under discussion, abundance of water in the deposit requires the installation of a pumping plant.

The method described should not be employed in deposits more than approximately 3 metres thick, since the arrangement of lagged stulls to hold the fill would then be dufficult. True, underhand stoping is sometimes also used in working thick ore bodies, but then only with very firm rocks and in unsupported stopes (see Fig. 351 below).

CHAPTER XXI

# METHODS OF MINING THICK ORE DEPOSITS

## 1. Particular Importance of Enclosing Rock Control in Working Thick Ore Deposits

In a thick deposit the size of mined-out space becomes correspondingly greater. If, on top of this, the shape of the ore body is irregular, the contours of stoped-out space likewise assume a rather complex outline. In this case *control of enclosing rocks* in working thick deposits becomes a matter of paramount importance.

Common in ore mining usage is the expression "support of excavations", which is synonymous with "control of enclosing country rocks". The first of these terms, however, can hardly be regarded as sufficiently apt, since there exist, for example, methods of mining of a very high practical importance (described below) whose basic principle is that the excavations *are not supported at all* because the ground capping the deposit or the ore itself are systematically allowed to fall.

Methods employed in the control of enclosing rocks during the mining of thick deposits play such an outstanding part that, as we shall see in Chapter XXIV, they usually serve as a basis for the classification of systems employed in mining ore bodies. Therefore, the description of the principal systems applied in mining wide veins and thick ore bodies in general is made along the lines based on the modes of enclosing rock control: 1) leaving natural support pillars, temporarily or completely abandoned; 2) artificial support; 3) filling; 4) shrinkage-stoping of ore; 5) caving of cover rocks; 6) caving of ore, induced or spontaneous; 7) the combination of these methods.

## 2. Mining with Natural Support Pillars

A very simple method of controlling enclosing rock in the space surrounding the excavation is by leaving support pillars containing the mineral. If such pillars are of appropriate size, the country rocks do not displace noticeably.

The support pillars in this instance can be left at random, in thinner sections of a deposit or at the sites of the occurrence of lean ores, if both are made possible by the geological structure of the deposit.

We have already had occasion to discuss an example of mining with the natural support pillars depicted in Fig. 50 (Chapter III). Now we shall list some data on the actual extraction of ore. Thick flat-dipping deposits of impregnation copper ores, occurring in strong rocks of sedimentary origin, are mined in Dzhezkazgan, Kazakh Republic. The back is supported not only by abandoned ore pillars (Fig. 348) but also by the rock masses surrounding the ore bodies, for their width is relatively small. Except for natural pillars, the excavations have no other support.

Actual mining is done by blasting in *underhand stopes*. The ore is hauled by scrapers. Because of underhand stoping, output per faceman per shift is rather low—around 2 cu m. Although ore pillars are abandoned, the ore losses are relatively low—about 15 per cent, this being due to the strength of ore and firmness of country rocks.

The mode of stoping, shown in Fig. 348, has one substantial disadvantage: the scrapers, depending on the site where they have to handle the ore across the stope, must travel in different directions. In recent years, consequently, a modification of the same system has been introduced (Fig. 349). It involves the use of central *working trench 1*, which is driven, prior to stoping, to a width of 3 metres up the entire height of the deposit by the overhand method with shrinkage-stoping. Surplus ore is slushed by scrapers into ore passes *3*. The bulk of the ore is drawn from underhand stopes *2*, with the broken ore falling into the working trench, and is then hauled to ore passes. Circular support pillars *4* are set up simultaneously with extraction in a manner making most of the ore obtained during the breakage fall into the central trench too. The rest of the ore getting into the previously mined section is carried away by scrapers during the ultimate clean-up of the worked-out area. This modification of mining with a central working trench ensures a large front of working faces and uniform direction of scraper travel. However, the system also has important drawbacks: a considerable time is required for the development of rooms for stoping, control over the state of the back along the full length of rooms is difficult, and passage of men and delivery of materials on the benches are rather inconvenient.

The ore in rooms can also be mined by shooting blast-holes (Fig.350). From ore passes *1* break-throughs *2* are driven on the level of the room floor and from these raises *3* are put up. All these workings run in the vertical plane along the axis of the projected room. The raises *3*, *3* are connected by break-through *4*, driven near the roof of the future room. The break-throughs of the adjacent rooms are connected by manway *5*. Stoping is started by "cutting" a room across its full span near the roof, to a height of 2.5-3 metres. To speed up the work, the cutting is done by two headings advancing from the opposite ends of the room towards the centre. The ore in the cutting is mined

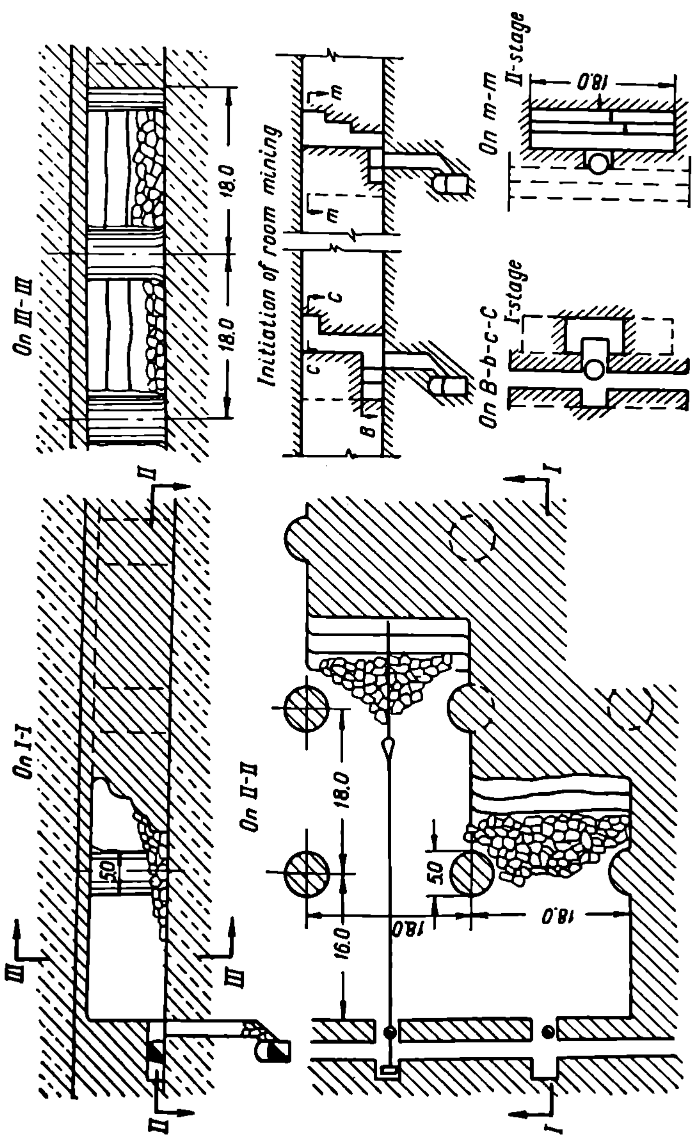



*Fig. 348.* Continuous breast-stoping with natural support pillars

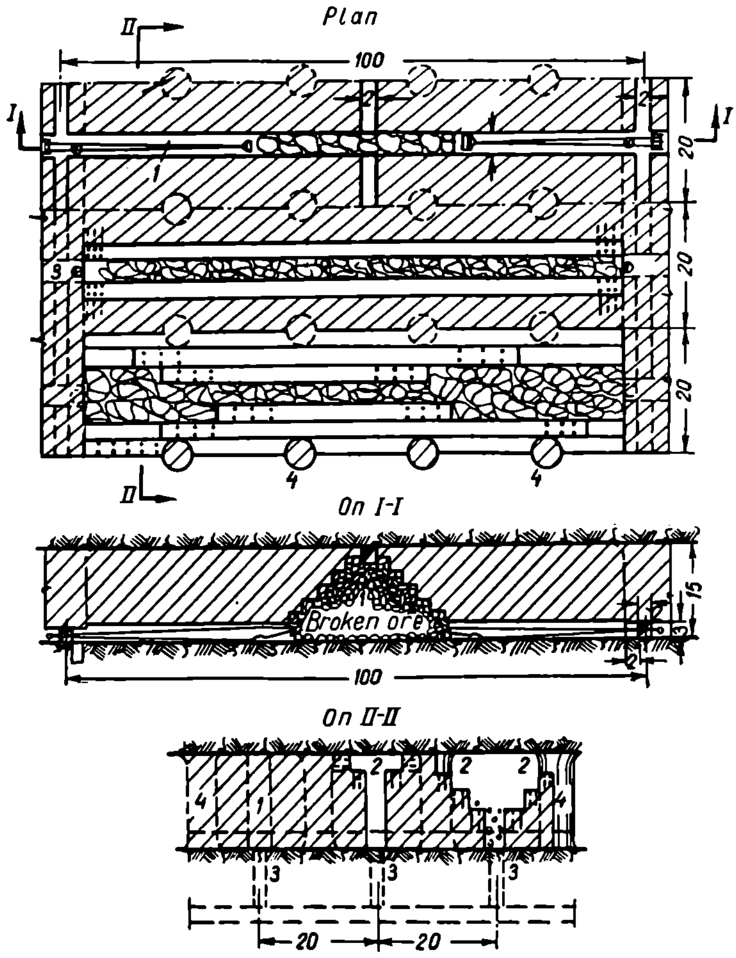



*Fig. 349.* Mining with a central working trench

by blasting of ordinary holes. On the completion of the cutting job over the entire area of the room, further stoping is done in just one bench *6*, the ore being broken by vertical deep blast-holes. Communications between stopes are then maintained by manway *5*. The broken ore is hauled by scrapers. The principal parameters of the system are shown in Fig. 350.

The method just described is sometimes designated as *room mining*. This term is not quite to the point, for in this case there are no typical "rooms". Nor would it be right to call it room-and-pillar mining,

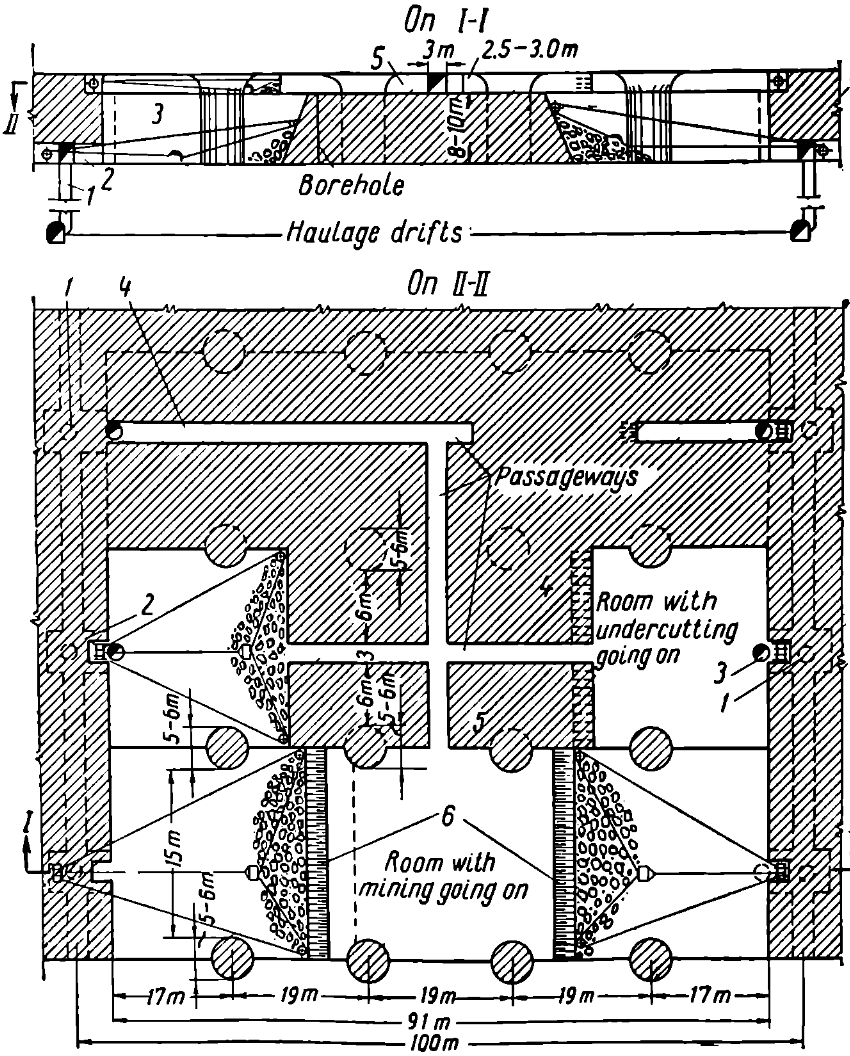



*Fig. 350.* Long hole breaking of ore in rooms

the name under which it sometimes goes, for it does not provide for the robbing of pillars. Hence, the method may be termed *mining with natural support pillars.*

In the working of thick ore deposits with flat-dipping occurrence involving the abandonment of support pillars, the mined-out areas are distinguished by their extensive size and absence of artificial support. In the U.S.A., in these conditions, use is made of highly efficient heavy equipment, such as loading machines, power shovels (of small capacity), drill jumboes with automobile and caterpillar drive (they were mentioned in Section 9 of Chapter XIX), bulldozers and large-capacity mine cars, both ordinary and self-propelling.

Fig. 351 depicts an instance involving mining of a flat-dipping deposit (angle of dip 10-15°) by a similar method. The ore is of medium strength with relative hardness of 6-8, according to Protodyakonov's scale. The bed is capped by firm limestone, allowing large exposures of the back. The bottom includes limestone too.

Because of the firm back, unsupported rooms have to be 12 metres wide. Natural support pillars measure 4×4 metres and are 4 metres apart. The length of the rooms is 60 metres. Ore losses in pillars come to 18 per cent of its total reserves. The ore from stopes is hauled by scrapers. The amount of explosives consumed per 1 cu m of ore and rock is 1.4 kg. Consumption of mine timber is quite insignificant—4 cu m per 1 000 cu m of ore mined. Output per faceman per shift is as high as 3.3 cu m.

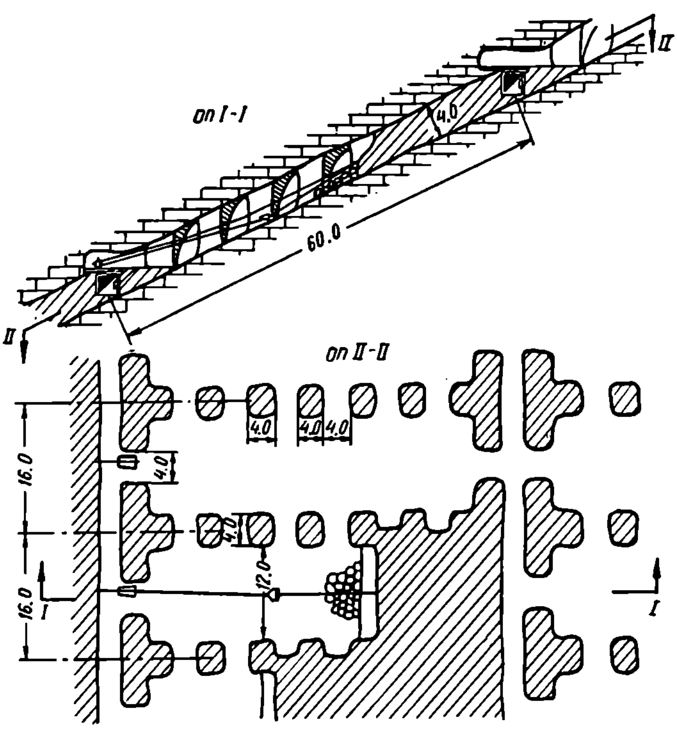



*Fig. 351.* Continuous breast-stoping with natural support pillars in a flat-dipping deposit

Methods of mining, like the ones described above, are sometimes included in the group of "open stope" or "open goaf" systems. The author, however, holds that these designations are not particularly suitable, inasmuch as the classification of methods employed in mining ore bodies according to the mode of wall-rock control provides, in this instance, for a specific feature, namely, the leaving of natural pillars to support the enclosing country rocks and not leaving of the worked-out area without support.

### 3. Sublevel Stoping

The basic points underlying the method are explained by Fig. 352. The latter refers to the working of a steeply dipping pyrite lense extending over 70 metres on strike and about 40 metres down the dip, and of a thickness of 19 metres. Along the boundary of the ore body raises or "pull-holes" are carried up from the lower haulageway from which *subdrifts* are driven in the centre of the ore body every 10 metres (in plan) over the entire length of the lense on strike. The most outstanding feature of the method is the mode of ore breaking. This is done from the ends of the drifts adjacent to the room (on the left in Fig. 352). No men are engaged in the room itself. Cross headings are driven from the end of each subdrift to make possible the breaking of the ore up the entire thickness of the ore body. Only one side of these cross headings, however, opens into the room, and that is why they are commonly designated as "slabs" or "open-end cuts". They accommodate drillers and drilling equipment. To prevent them from falling, the men here are provided with safety belts. The holes are drilled both from these slabs and directly from the ends of drifts upwards and downwards. The blasted ore drops into a chamber whose bottom is shaped like a *funnel-like opening* with discharge chutes below.

The lag of the stoping operations in the upper sublevel behind those in the underlying ones, shown in Fig. 352, that is, the presence of unbroken ore hanging over the room, is permissible only in the case of very strong, firm and jointless ores.

The example of sublevel stoping shown in Fig. 352 deals with an exceptional case, where the size and the shape of the ore body allow its mining with a single room or chamber, the ore being broken from sublevel drifts. If, on the other hand, the deposit is more extensive, it has to be preliminarily divided by development openings into separate extraction blocks. One example of this is the modification of the system illustrated in Fig. 353 where, to widen the front of working faces, the block is extracted in two directions from the central raise.

Let us consider more closely this latter modification of the method.

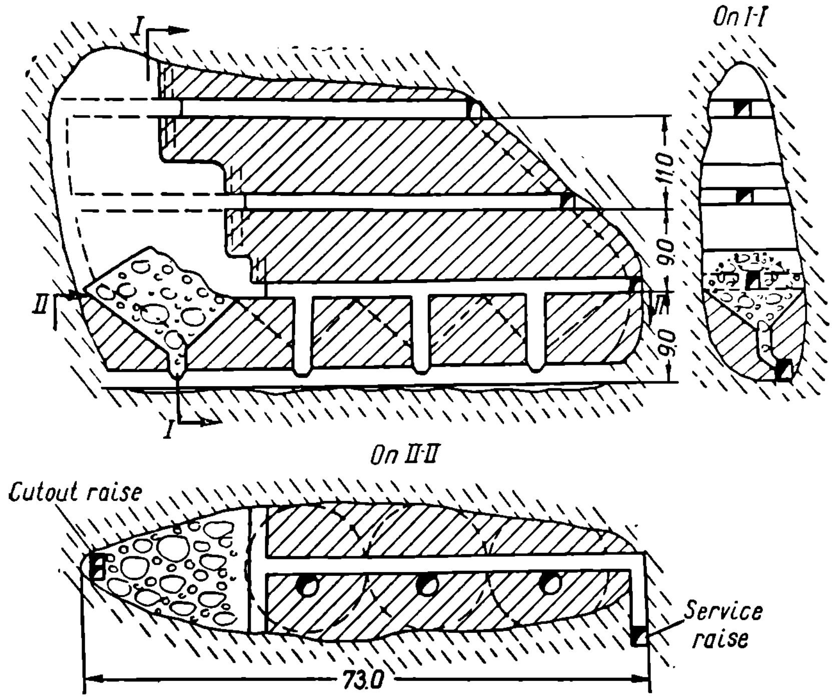



*Fig. 352.* Mining of a single pyrite lense with the breaking of ore from sublevel drifts

The lateral width of the ore deposit comes to 12-20 metres, its angle of dip is 50°. The vertical level interval is 56 metres. Two rooms *1*, *1* are mined in the block, each being 30 metres long. Rib or interchamber pillar *3* is 8 metres wide. Uphill opening (raise) *4* is driven along pillar *3* axis, near the foot wall, to connect the haulage and airway levels and to ensure the progress of development work.

From lower haulage drift *1*, cut in ore, ore passes *2* are arranged every 6 metres all the way to the scraper or secondary breaking level. The bottom of the room is widened to form funnel-shaped openings to receive the broken ore.

From working *4* cross drifts (crosscuts) are driven and from these sublevel drifts *5*, at vertical intervals of 9-11 metres.

The general direction of ore extraction is from the block boundaries towards its centre. The ore is broken from side-cuts *6*.

Upon the extraction of the ore inside the rooms (the so-called *room stocks or reserves*) the rib pillar is destroyed by explosives filling holes *7* in it (see section *CC* in Fig. 353). Lastly to knock down the floor pillar or "ceiling", deep horizontal blast-holes *8* are first drilled above

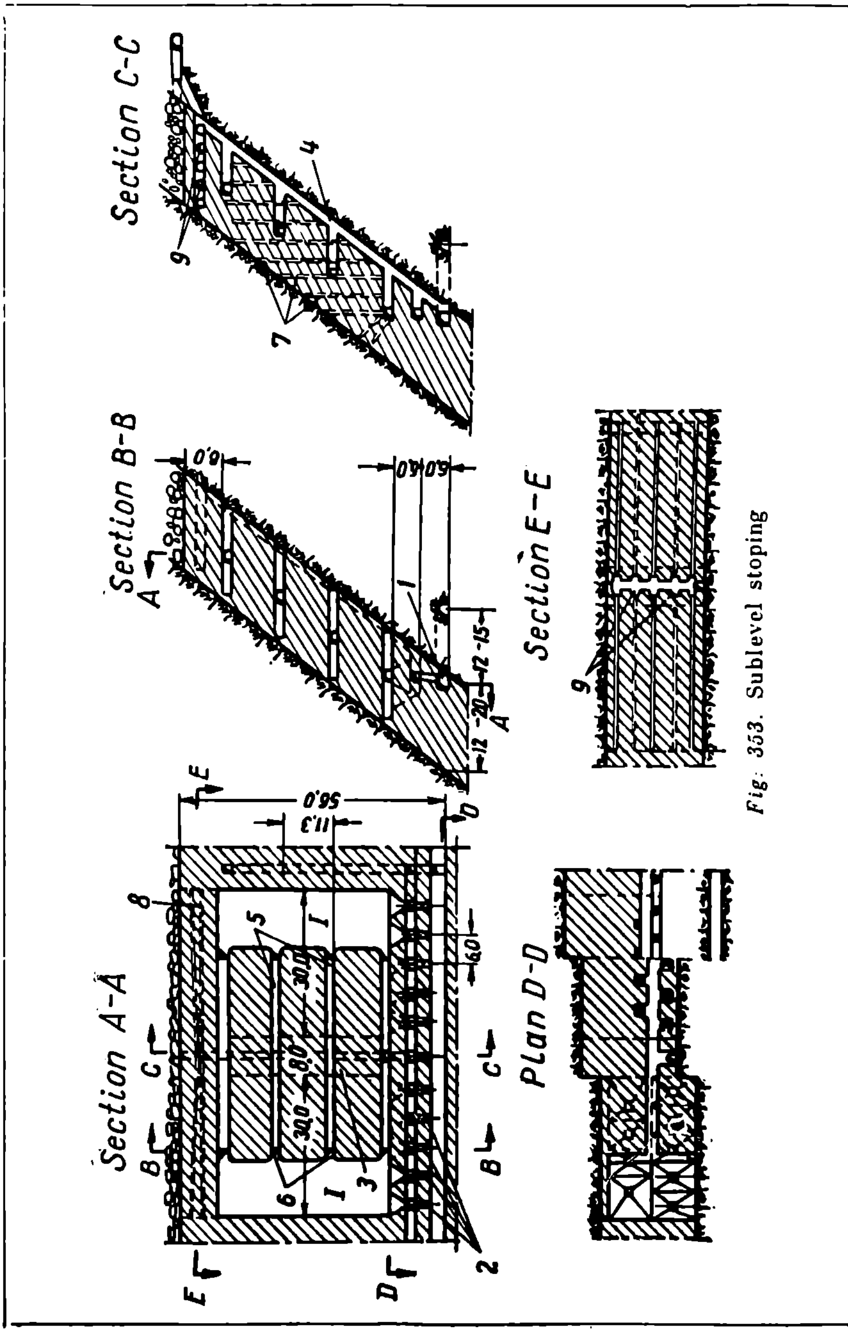



Fig. 353. Sublevel stoping

the rooms. These are made from special drill chambers *9* in the ceiling over the rib pillar.

If any sill pillars had been left above the drift on the air horizon above the ceiling, they are shot down together with the ceiling.

Since it is in the room that the mining is most productive, the size of the room, as well as that of the rib pillars, ceiling and sill pillars should be so chosen that they contain the biggest possible ore *stocks in the room*. But this depends on the degree of the stability of ore and enclosing rocks.

Figs 352 and 353 refer to an instance when the general direction of mining is on strike. But if the ore body is of lateral width in excess of approximately 20 metres, the axes of the rooms mined from sublevel drifts—in this case crosscuts—run across the strike (Fig. 353).

Depending on local conditions, the ore can be excavated by the various methods indicated in Fig. 354:

1) holes 2-2.5 metres long are drilled up and down when the general line of working faces extends vertically. The vertical subleve- distance in this instance should not exceed 5-8 metres;

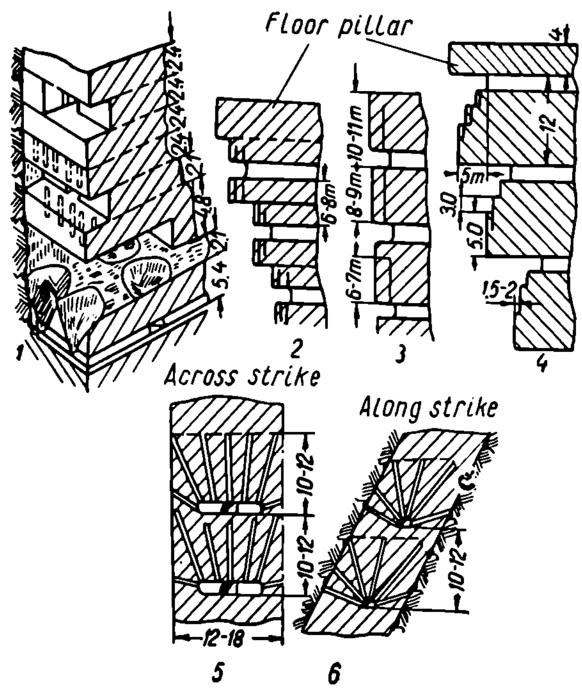



*Fig. 354.* Methods of ore breaking

2) the type of holes are the same, but the faces are established in an overhand fashion, which is possible only in very strong ore;

3) long-hole drilling with sectional steel in a sublevel stope. This makes it possible to augment the vertical sublevel interval to 10-15 metres and more. In individual cases—for instance, at the Vysokogorsk mine in the Urals—the depth of blast-holes with the method of mining under discussion comes to 45 metres.

One disadvantage of the method is that the ore may break off in big blocks (the so-called oversize), complicating its drawing;

4) for this reason underhand stoping is possible when ore is very strong;

5) to reduce the number of slab entries, the holes may also be drilled in fan-shaped order.

Long-hole rounds have a substantial advantage, for they make it possible to increase the vertical sublevel interval, that is, cut down the volume of development work and enhance the efficiency of drilling (from 40-50 to 60-70 tons with shallow-hole round and from 80-100 to 150 tons with sectional steel per man per shift).

The broken ore drops down to the bottom of the room. Its immediate discharge through the chutes is possible only when the ore is broken into pieces of adequate size. Otherwise, a *breaking* or *grizzly level* has to be provided for. An example of how the grizzlies are arranged is given below.

Scraping can be used for conveying the ore to the main loading station.

When the room stocks have been extracted, the mining of ore from floor and rib pillars and the sill pillars over the drifts follows along two basic lines: 1) by preliminarily filling the stoped-out area in the rooms with waste, or by transferring the latter from the overlying sublevels, and 2) with unfilled rooms.

In filled rooms, the pillars can be recovered by the methods of top slicing discussed below (Section 8) or sublevel caving (Section 9).

If the room is not stowed with filling material or waste, the floor and rib pillars may be destroyed by explosive charges. One of the methods of such *shooting* is explained by Fig. 355. This refers to the mining of a very thick ore body and for this reason the 16-metre-wide rooms run across the strike. The floor and rib pillars are shot down by deep blast-holes drilled in the ceiling in fan-shaped rounds to obviate driving additional workings to accommodate drilling machines and their shifting after the completion of each hole. When the ore obtained following the mass blasting of the floor and rib pillars and the spontaneous caving of the ground has been drawn, the sill pillar over the main level is usually robbed by sublevel caving (Section 9). A substantial drawback of the recovery of floor and rib pillars in an unfilled room is the high loss of ore and its dilution during the drawing process.

Sublevel stoping has a number of important advantages, such as: 1) safety of work, for the men are engaged not in rooms, but in workings of a small section and are supplied with safety belts tied to a rope; 2) high efficiency of ore breaking; 3) wide general front of working faces; 4) high tonnages yielded by one production block; 5) no timbering in the rooms; 6) delivery of ore to pull-holes by gravity.

On the other hand, the system has some serious shortcomings: 1) driving of a large number of development and subsidiary openings; 2) a complex method of pillar recovery; 3) large total losses of the mineral; 4) high losses and dilution of ore during the robbing of pillars; 5) the waste in the stope cannot be separated from the ore and the ore is not delivered according to grades; 6) the hazard of the spontaneous caving of the ceiling and wall rocks when the parameters of the system are inadequate.

According to Agoshkov, "the most propitious conditions for the application of this system are: high dip, thickness from 10 to 15 metres; medium-hard and strong ore of relatively low value with no

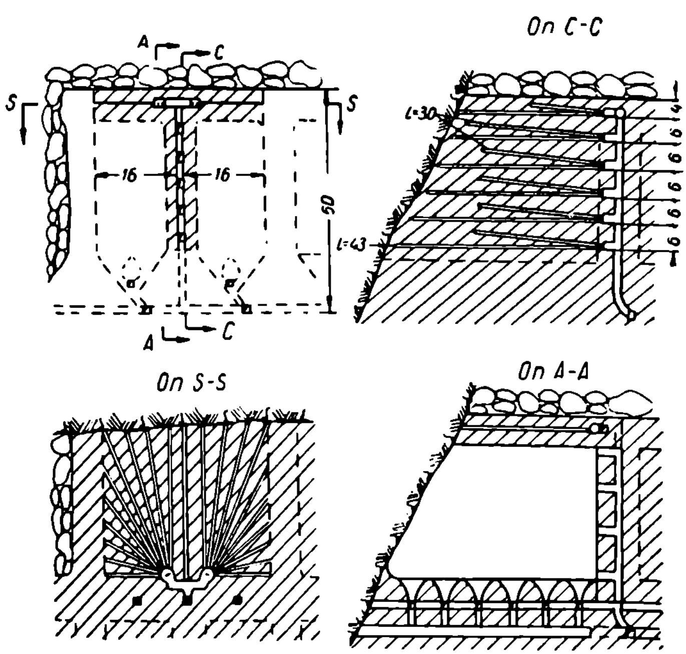



*Fig. 355.* Shooting down rib and floor pillars by long holing

inclusions of gangue; compact and firm wall rocks". The level interval usually ranges from 45 to 80 metres.

Sublevel stoping is employed very widely, especially in mining iron ore deposits.

### 4. Breaking of Ore by Deep Horizontal Blast-Holes and Coyote or Tunnel Blasting

There are methods of mining generally analogous to sublevel stoping, except that the breaking of room stocks of ore is effected by firing explosive charges in deep horizontal blast-holes or by coyote or tunnel blasting.

The first of these methods is shown in Fig. 356, depicting the mining of a thick steeply dipping occurrence of iron ore. The level interval is 60 metres, the width of the rooms—30 metres. The ore is broken by a series of horizontal blast-holes bored by drilling machines set up in special "drill chambers" *2*. The pattern of hole rounds is similar to the one shown in Fig. 373 (see below). Development openings are made in rib pillars or in foot wall *3*. Temporarily abandoned pillar *1*, near the hanging wall, is recovered later, before the caving of the ceiling.

If the ores are very strong and viscid, use can be made of the highly efficient method of mining which involves breaking the ore by concentrated (*coyote*) explosive charges (Fig. 357).

Development and subsidiary work in the block is as follows.

Lateral foot wall drift *1* and ore drift *2* are run on the haulage level. Block raises *3* are put up to the subjacent horizon along the axis

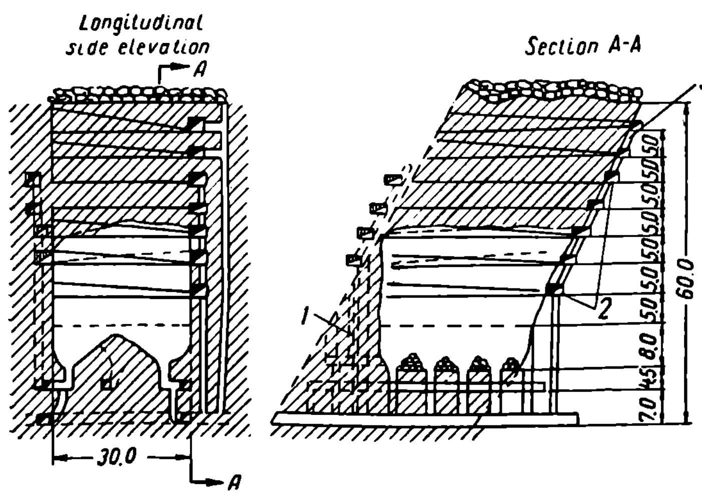



*Fig. 356.* Breaking of ore by deep horizontal blast-holes

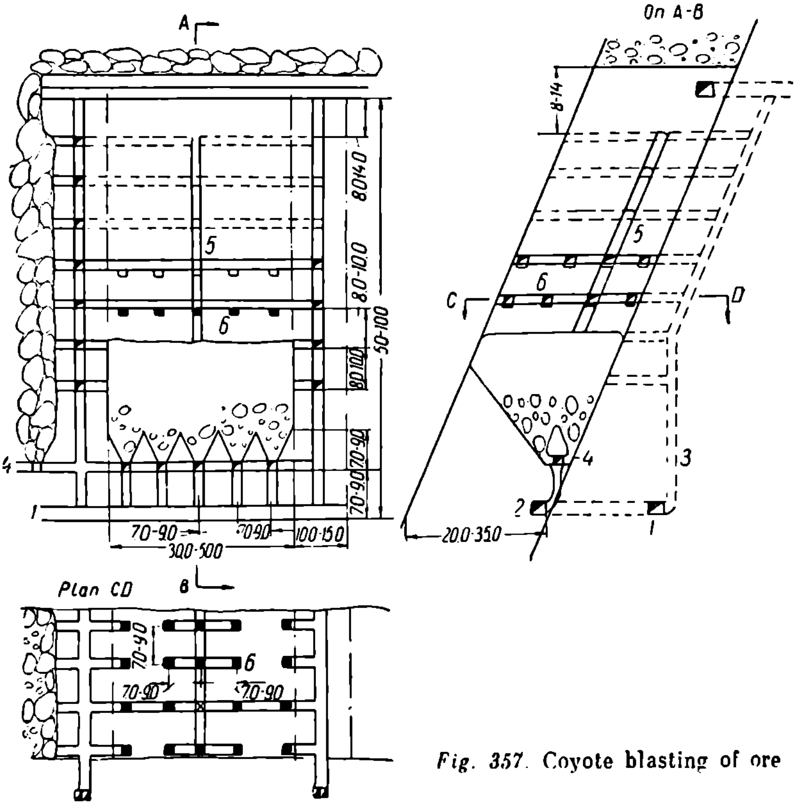



*Fig. 357.* Coyote blasting of ore

of rib pillars. At a height of 7-9 metres above the haulage level drift 4 of the secondary or grizzly horizon is run and it is connected with ore-pass raises which open into the haulage drift and are put up every 8-10 metres.

From the grizzly level twin inclined dump chutes are driven at an interval of 8-10 metres to the height of 7-9 metres, their top portions extended. Cut-out raise *5* is put up the full level interval from the grizzly level drift (usually along the axis of the room).

From the raises in pillars and along the axis of the room powder drifts *6* are run in the plans of each sublevel. The sublevel interval (the line of least resistance ) is generally 8-10 metres. The section of powder drifts is kept as small as possible, usually $1.5 \times 1.5$ or $1.5 \times \times 1.8$ metres.

When these openings are driven, only 50 per cent of the ore is cleaned up, the rest being left in the workings to be used for stemming tunnel charges. The layout of powder drifts and crosscuts should ensure uniform distribution of explosive charges in the solid mass to be blasted and a more uniform size grading of broken ore.

Stoping involves breaking ore within the bounds of the room.

In the case of sectional breaking, a slot 18-25 metres wide and as high as the room is first formed by coyote blasting. After this the ore in lateral sections (up to 15-20 metres wide) is coyote-blasted on both sides of the cut-off slot.

In the case of slice breaking, no cut-off or opening slot is made, the ore within the sublevel being broken simultaneously all over the room.

For coyote blasting ammonites Nos. 6-7 are employed.

After being charged the powder drifts and crosscuts are stemmed with ore left over from the driving of these openings. In order to prevent the destruction of other workings in the production block by the air wave, the site of coyote blasting is tightly sealed off.

In tunnel blasting, concentrated charges are fired simultaneously with the aid of a detonating fuse and electric caps.

Ammonite consumption in primary breaking for the mine as a whole averages 0.8-0.9 kg per ton. Coyote blasting, however, yields a considerable number of large ore blocks, requiring a great deal of secondary breaking or block-holing. The amount of explosives consumed in secondary breaking is as high as 0.1-0.25 kg per ton and more, and that constitutes one of the major disadvantages of coyote blasting.

The 8-10-metre-thick floor pillar is also shot down by coyote blasting, the concentrated charges being placed in openings made earlier.

Quite often the floor pillar is shot down and the last ore slice in the room is broken simultaneously.

Interchamber or rib pillars are recovered either by employing the "room over pull holes" variant, in thicker deposits, or by coyote blasting simultaneously with the floor pillar.

Output per faceman per shift is 20-22 tons, while the tonnage produced by one driller for the entire block per shift may be as high as 80-100 tons.

Factual ore losses reach 8-12 per cent and dilution amounts to 4-6 per cent.

The typical parameters for this method of mining are shown in Fig. 357.

## 5. Square-Set Stoping

The extensive worked-out areas appearing after the extraction of thick deposits cannot be supported by ordinary posts or stull sets. Therefore, mine timber employed for supporting stopes in such ore bodies is framed into spatial square sets.

The main idea behind the construction of *square-set support* in a stope is to set up mine timbering in the form of space lattice made of round sticks or bars. The elements (Fig. 358) of such a lattice are 12 units placed along the edges of a parallelepiped. Two vertical (posts) and four horizontal members meet in each joint of the lattice, with the two horizontal running parallel to the stope usually called *caps* and the two perpendicular *braces*. The size of the square-set members depends on the expected rock pressure and varies widely, while the height of the posts in the clear of the set usually comes to 1.8-2.5 metres, the length of caps to 1.5-1.8 metres and that of braces to 1.2-1.8 metres.

It is very important for the members of a square set to fit together perfectly.

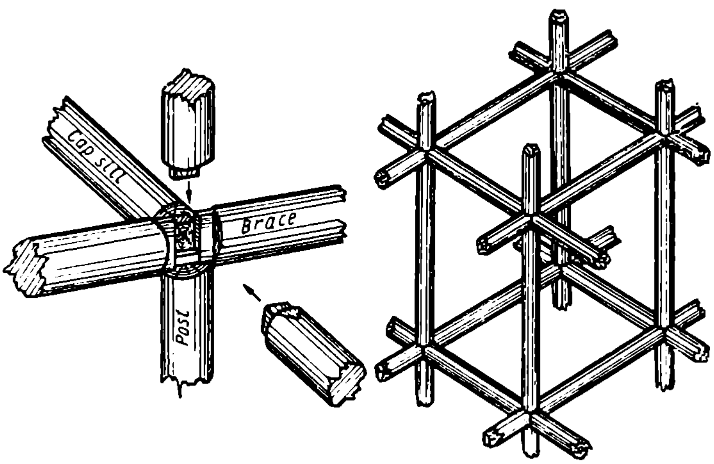


*Fig. 358.* Square sets

The units of square sets can obviously be used to fill a space of any shape and size (Fig. 359). To make the square set stable, its upper members, lying for the time being near the surface of an unblasted solid mass of the mineral or wall rocks, are reinforced by knee braces, wedges, temporary posts and stulls. The horizontal members or girts of the square sets are covered with provisional flooring for the workers to stand on, and to transport mineral, filling materials, etc. The broken ore is lowered to the haulageway along the dumping slopes arranged in square sets.

The mined-out space supported by square sets may remain unfilled or be stowed with waste. Experience shows, however, that although the adjacent members of square sets should hold each other in the definite position they are made to assume when they are set up, a square set as a whole is a very "delicate" structure, which is easily disarranged by heavy rock pressure and consequently loses its capacity of withstanding the weight of the ground. Square sets with no

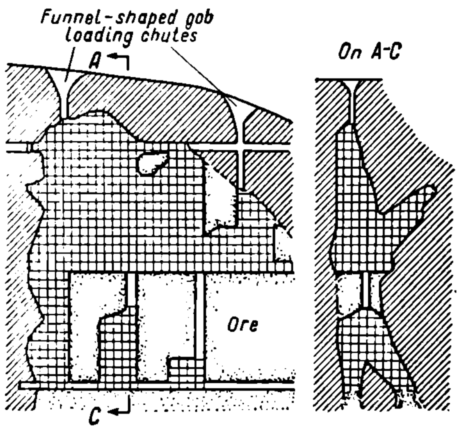



*Fig. 359.* Square-set stoping

waste filling are, therefore, employed only in working ore deposits containing particularly firm ground and then only serve as a scaffolding. If used otherwise, the sets are filled with waste to make them sufficiently strong. When any individual members of the set become displaced it must be reinforced immediately by additional posts, stulls, diagonal braces and even cribs.

The availability of square sets does not eliminate the danger of ore cavings in the course of the undercutting of extensive areas of the ore body. Therefore, the square-set stoping of the ore—naturally done from bottom up—should involve the undercutting of a small area of the ore body. Hence the mining of large steeply dipping ore deposits with square sets is effected in *vertical sections* 4.5 metres wide of a length equal to the lateral width of the ore body. Such working sections are called *long* working sections or cuts. When the ore is weak, the area of these working sections may be limited not only along the strike, but across it (*short working sections* or *cuts*).

If practised by vertical sections or cuts, square-set stoping with filling helps minimise ore losses even in working ores and rocks of medium stability. Moreover, it creates favourable conditions for sorting ore and leaving the gangue picked out in the mass of the mine-fill. On the other hand, this method is distinguished by low labour efficiency and high consumption of mine timber, and for that reason it should be employed advantageously only in working valuable ores.

## 6. Cut-and-Fill Method of Mining Thick Ore Deposits

It depends on the structure of an ore body whether the filling material is obtained from the country rocks enclosing the ore or, in the case of monolith ores, is supplied from outside via the upper drift.

For example, Fig. 360 is illustrative of mining a complex silver-lead (plumbagine) vein, whose extraction yields a large amount of barren ground which is subsequently used as filling material for stowing worked-out space. If there is a surplus of gangue, it can be lowered through dumping chutes (or ore chutes) to the horizon below. Rich ore is excavated separately (selectively) and passed down ore chutes which are built of large lumps in the midst of the mine-fill and are gradually extended.

Broken ore is conveyed by dumping chutes to the lower level (Fig. 361). The filling material is supplied to the stope from the upper drift and spread out with the aid of shovels or scrapers. The drawing depicts a stepped stope with the holes made in solid ore by a column-mounted drilling machine.

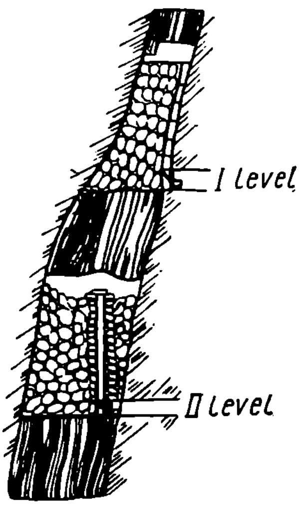



*Fig. 360.* Mining of a high complex shoot with vein-rock filling

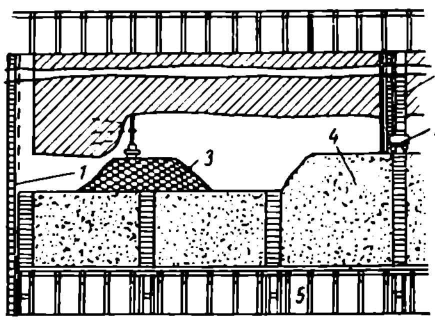



*Fig. 361.* Cut-and-fill stoping
1—ladder compartment; 2—waste-fill chute; 3—broken ore; 4—mine-fill; 5—ore-discharge chute

Here are two more examples of cut-and-fill stoping of large ore deposits. Fig. 362 illustrates mining a highly dipping sulphide ore deposit about 6 metres thick. The ore is jointed and susceptible to exfoliation and formation of loose slabs. The level interval is 45 metres and the length of the block on strike comes to 25 metres. Since the filling material is supplied from the upper drift through the fill compartment of one of development openings bordering on the extraction block, the surface of the mine-fill is made inclined to facilitate its distribution in mined-out space. The flooring is also used for the passage to the chutes of the ore broken in the inclined stope.

The raises delimiting the extraction block and serving as a passageway for the fill are put up in the hanging wall of the deposit, and the chutes are cut in its foot wall. Filling eliminates the need for sill pillars over the haulageway and protective pillars near the raises. This cuts ore losses to 2-3 per cent.

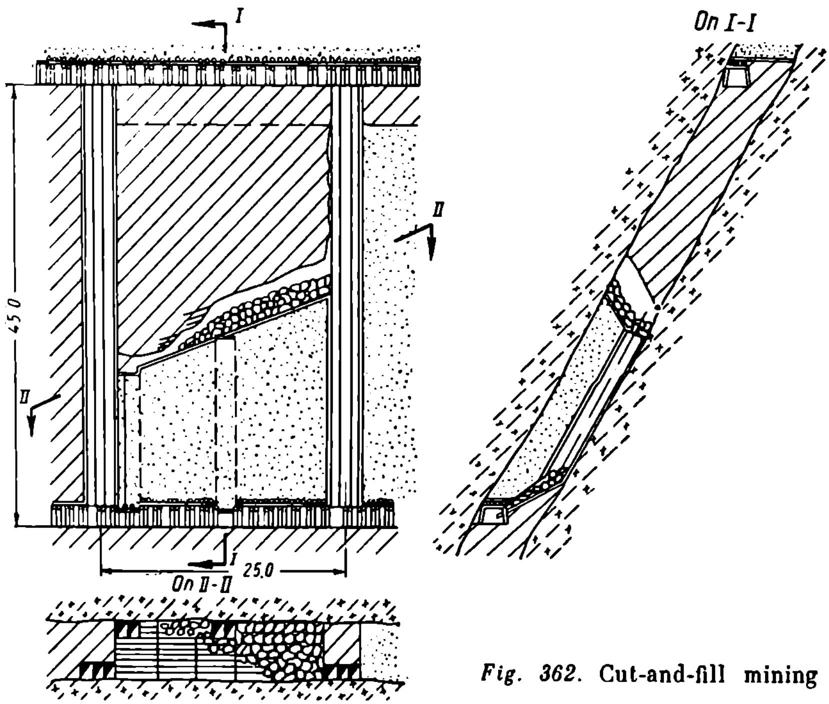



*Fig. 362.* Cut-and-fill mining

Fig. 363 illustrates the cut-and-fill stoping of a highly dipping ore body of an average thickness of 25 metres. The ore is excavated in rooms which are stowed immediately behind the advancing stopes with filling materials supplied from the air level. The rooms are made with a span of 12 metres, the width of the ore pillars temporarily left between them being 9 metres. As the ore body is of considerable thickness, the long axes of the rooms run across the strike. Ore chutes are arranged in the mass of the fill and have timber crib lining. Two rows of ore chutes are put up along the room, at a distance of 6 metres between their axes. The raises are put up at the edge of each intermediary (rib) pillar. They serve as manways and for ventilation. Later, in the process of robbing pillars lying between rooms stowed with fill, they play the part of development workings through which ore is recovered from pillars by methods described below, that is, by horizontal top slicing and sublevel caving.

Cut-and-fill stoping of ore deposits reduces ore losses to the minimum and, therefore, it should be applied in mining valuable ores. The

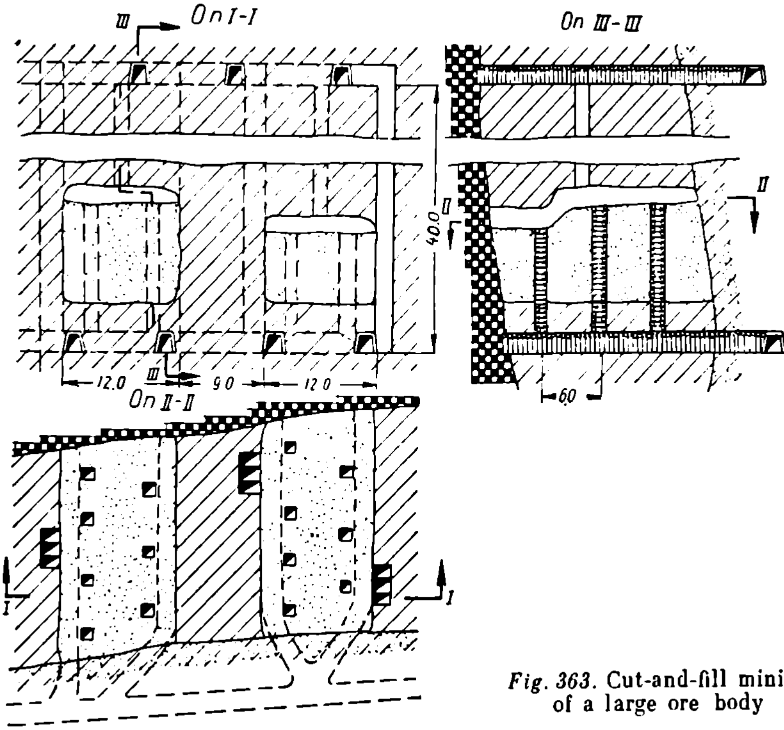



*Fig. 363.* Cut-and-fill mining of a large ore body

system is particularly suitable for working deposits of self-igniting ores (pyrite ores). There is a growing tendency towards wider use of hydraulic fill (filling by flushing).

## 7. Shrinkage-Stoping

The preceding chapter described *shrinkage-stoping* as applied in mining thin and medium-thick ore bodies. The same principle of country-rock control is also widely used in working large ore deposits.

Here are two typical examples. Fig. 364 depicts mining a sulphide ore deposit 5 metres thick, dipping at an angle of 65°. The deposit is cut into extraction blocks by raises put up every 25 metres, with a vertical level interval of 45 metres. The sill ore pillars left over the haulage drift are 4.5 metres wide and at most 7 metres high. The upper part of the pillars is chamfered to form pull-holes for the ore. The position of the working face and shrinkage-stoped ore is shown in the drawing. Small chambers are arranged above the drift, for the secondary breaking of oversize ore drawn from shrinkage-stopes. After

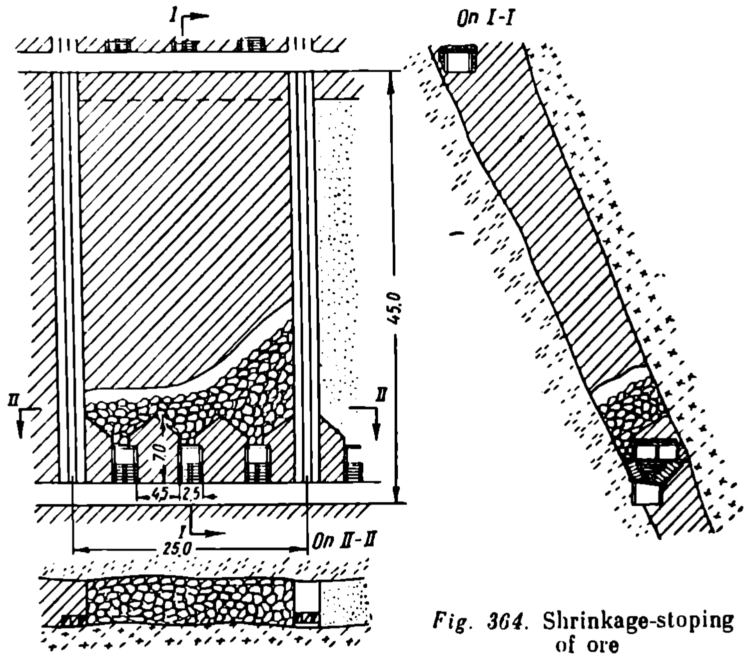



*Fig. 364.* Shrinkage-stoping of ore

the ore has been drawn off, the mined-out area is stowed with fill supplied from the upper drift. The use of filling minimises ore losses (3-5 per cent) and ensures low consumption of mine timber. The efficiency of facemen is quite satisfactory.

The layout of raises flanking the extraction block, as shown in Fig. 364, is admissible only in an ore body of moderate thickness and good-quality fill.

In more complex conditions, the raises are made in rib pillars, as depicted in Fig. 365, which illustrates shrinkage-stoping of an extensive ore body shaped like a dike made of igneous rocks with impregnation ores. The dike dips almost vertically and its thickness ranges between 9 and 30 metres. The ore-bearing and country rocks are very strong.

The vertical level interval, which is not divided into subfloors, is 50 metres. The shrinkage-stoped rooms are very extensive, exceeding 40 metres in height, the span (along the strike of the ore body) is 20 metres and the length equals the thickness of the deposit. The pillars between shrinkage-stopes are 6 metres thick. In these pillars rise headings are carried up, and from the headings crosscuts are driven every 6 metres along the height and are connected with shrinkage-rooms by short break-throughs. These openings are used for the pas-

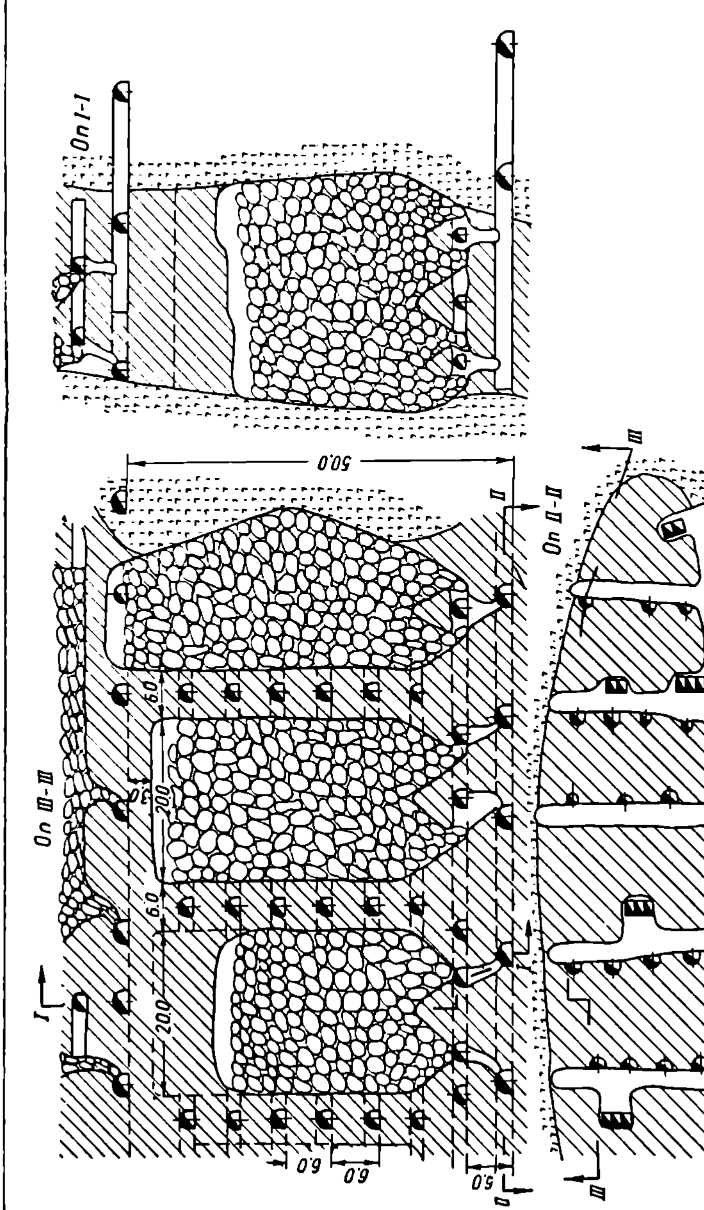



*Fig. 365.* Mining a very thick deposit by the shrinkage-stoping method

sage of men, air supply and the delivery of drilling equipment from the lower drift.

As usual, the distance between the face of the shrinkage-stope and the surface of the waste fill is maintained by drawing off the ore to leave sufficient working space for handling stopers. In other words, it should be somewhat higher than the man. Because of the extensiveness of shrinkage-rooms and hardness of ore, breakage may yield large blocks. For this reason a *grizzly level* is provided under the bottom of shrinkage-stopes (the bottom is in the shape of a series of funnel-like holes) and above the haulage horizon, where large lumps of ore drawn through hoppers (discharge holes) may be broken up into smaller pieces before being dumped into loading chutes (see Section 10 below). Since in this instance the long axis of shrinkage-rooms runs across the line of strike (on account of the thickness of the ore body), the loading chutes are not arranged directly in the haulage drift but open into crosscuts connected with it.

When an overlying level is approached, a *floor pillar* of ore 3 metres thick is left in the shrinkage-stopes.

Shrinkage-stoping makes it possible to recover floor pillars and pillars left near the haulageways and grizzly level of the overlying horizon. Just before the extraction of ore in the shrinkage-stope has been completed (see the right half of the vertical section on strike in Fig. 365) its face is carried up right through the floor pillar to the former upper haulageway. Numerous holes are drilled in the face of the shrinkage-stope in the top workings driven in the pillar adjacent to it, and their simultaneous shooting breaks the ore and adds it to that already stored in the stopes.

Ore tonnages yielded by this system of mining are rather considerable: output per man per shift is as high as 6 cu m; consumption of explosives is moderate although the ore is hard and comes to 0.75 kg/m$^3$, while that of mine timber is insignificant. The losses of ore are low (5 per cent), but its dilution is high (up to 15 per cent).

Shrinkage-stoping of large ore bodies is warranted only in the case of strong ore and firm country rocks. With these modifications of the method, when ore is drawn only at the bottom of shrinkage-stopes, selective extraction of ore according to its grades is impossible and ore should not have any sizable inclusions of gangue.

## 8. Caving Methods of Mining

The most typical case of working a large deposit by *caving cover rocks* involves *horizontal top slicing*.

The principle underlying this method is elucidated by Fig. 366, referring to mining a steeply dipping lenticular ore body up to 12 metres in thickness.

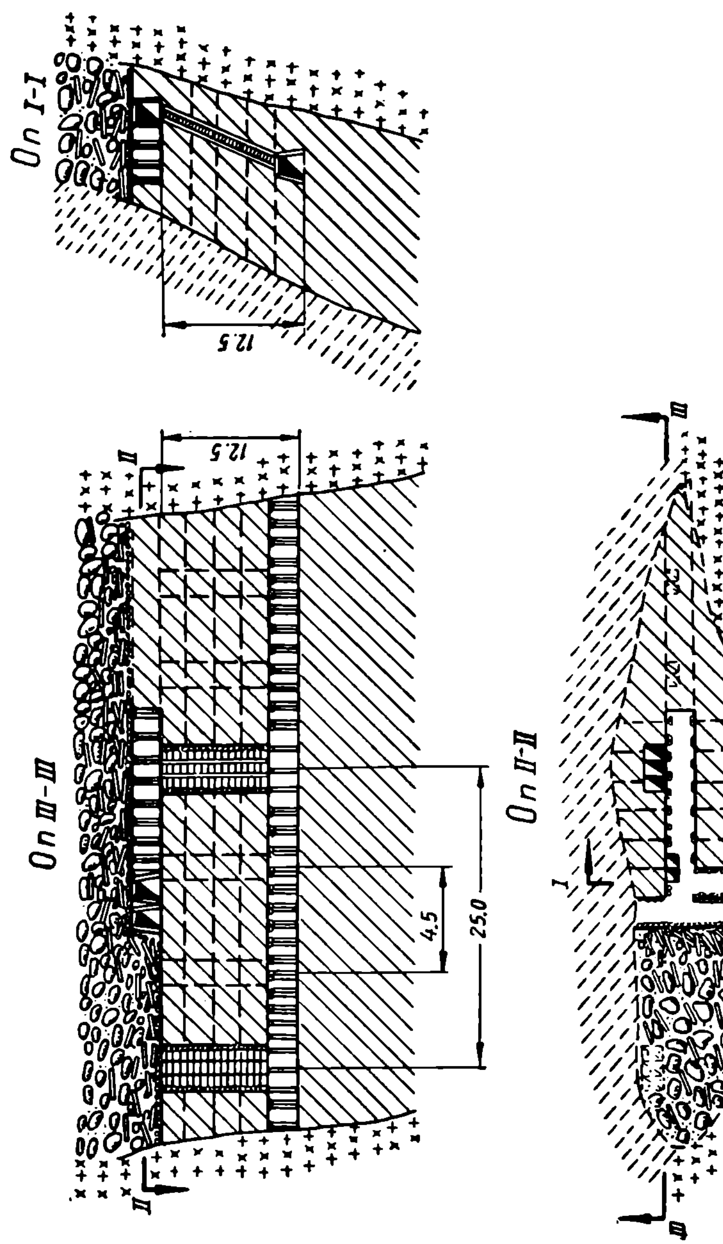



*Fig. 366.* Horizontal top slicing and cover caving

From a haulage drift made in the ore body inclined rise headings are carried up at 25-metre intervals on strike.

The ore body is extracted in descending order by horizontal slices 2.5 metres thick. To accomplish this, a slice drift is run from the rise heading along the strike of the lense intended for maintaining communications with the stopes and for the passage of ore via appropriate compartments. The ore in the slice is extracted on both sides of this drift by narrow slabs, generally running in a direction opposite to that of the corresponding ore chutes. Each slab cut should have very strong support, for the caving rocks exert appreciable pressure. When a regular slab cut has been extracted, timbering in the preceding slab is knocked down (shot down) and the ground is allowed to cave in over the entire area of the slab.

Ore in horizontal top slicing is usually hauled to discharge chutes by scrapers, although conveyers can be used too.

The use of scrapers to pull ore along the bottom of the slice presents some inconveniences and their movement is hampered by timber sets. To remedy this situation, special *scraper* or *storage trenches*, that is, trench-like recesses, are cut out in the floor of the slice to serve as a receptacle for broken ore. This scraper trench is used to slush ore to discharge chutes. Another method is to arrange *storage* or *scram* drifts *1*, driven every two or three slices (Fig. 367). They are connected with the working slices by small pull-holes *2* for the drawing of the ore, which is also hauled to the main discharge chute along the scram drift. Fig. 368 is illustrative of mining several horizontal slices instead of one.

To avoid contaminating the ore with waste, a timber mat is laid on the bottom of the topmost slice. As the slices are worked out, this mat gradually sinks and in each slice is covered by the pieces of timber which remain in the goaf after the timbering has been knocked down in the process of slice caving, and by new layers of the mat. This mat, separating ore from cover rocks, is called *flexible mat.* It not only prevents smaller pieces of ore in the back of the slice from falling through, but also helps even out the pressure bearing down on the face timbering. The system is characterised by the immense expenditure of mine timber—up to 100-150 cu m per 1,000 cu m of the ore mined. Other disadvantages are low output per man per shift and high consumption of explosives—usually 1.5-2 kg per 1 cu m of ore. The latter is attributed to the fact that the weight of ore proper does not help the explosive force. This method complicates ventilation of stopes. The storage or scram drifts described above also improve ventilation conditions in the stopes.

In order to enhance the efficiency of facemen, A. Gribin and N. Yenikeyev have proposed extracting slices not by slab cuts but by *long faces* (walls) with the use of conveyers.

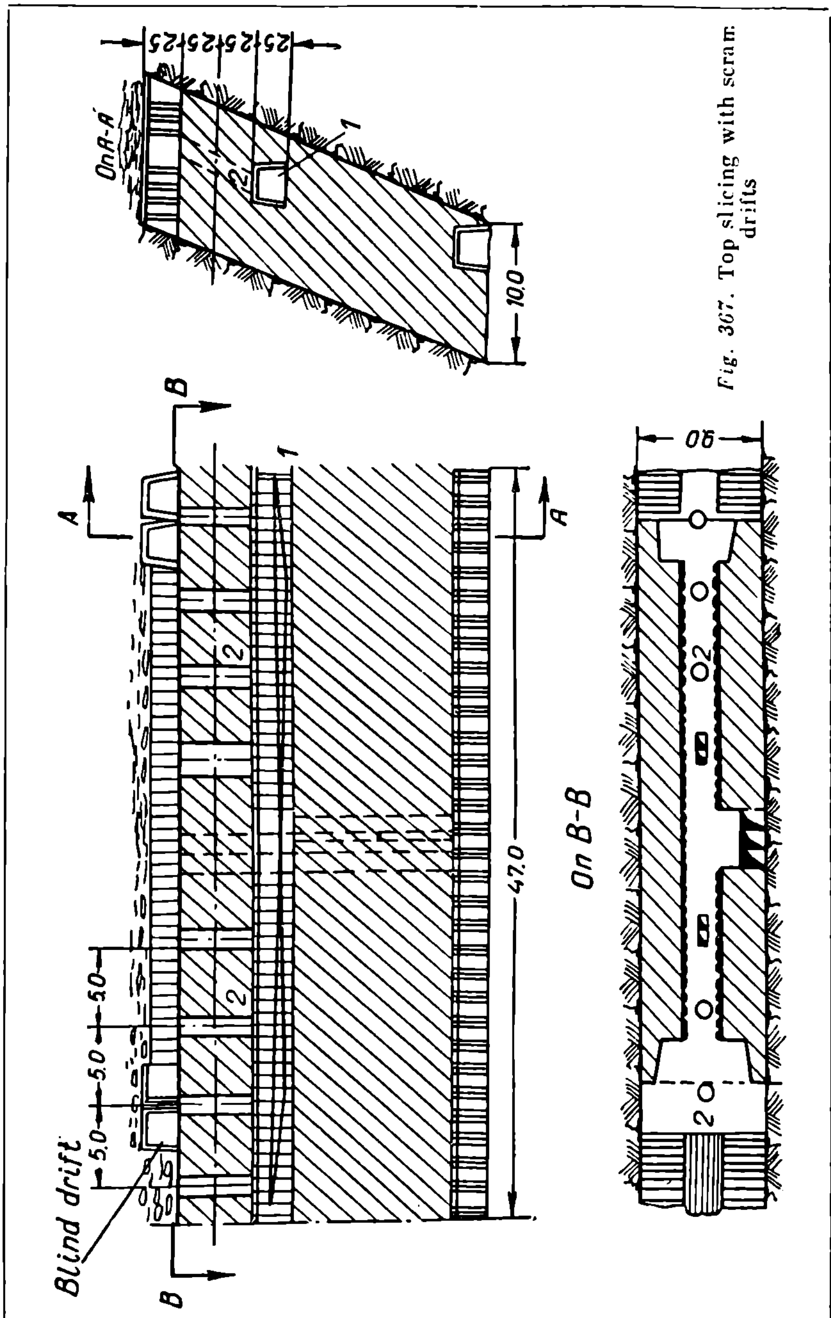



Fig. 367. Top slicing with scram drifts

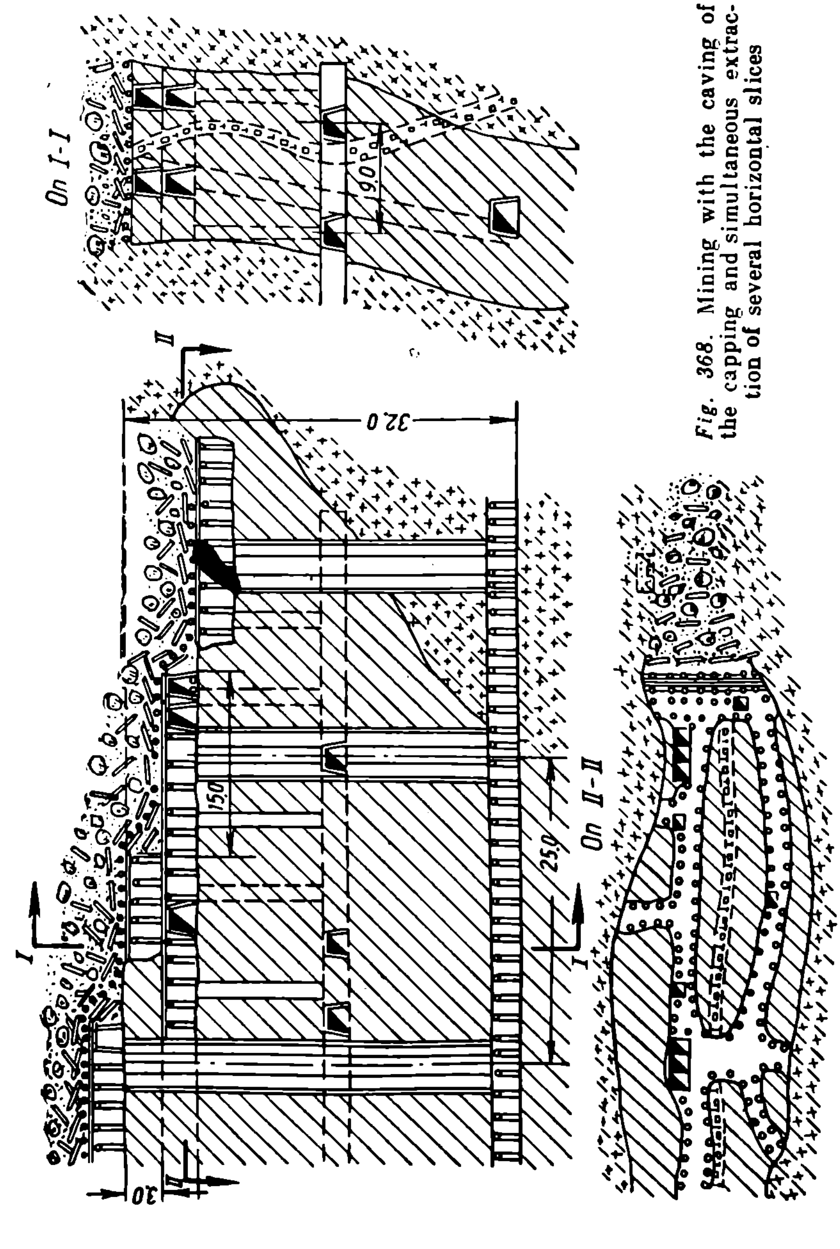



*Fig. 368.* Mining with the caving of the capping and simultaneous extraction of several horizontal slices

Among the positive aspects of the top-slicing method are the possibility of applying it for weak cover rocks and the insignificant ore losses (not more than 3-4 per cent when operations are properly conducted). The presence of firm country rocks and a strong ore makes this system unsuitable, since in this case the ground fails to cave immediately after extraction, while later there is a danger of spontaneous mass breakings. Horizontal top slicing may be employed in working extensive thick ore bodies of irregular shape, provided the layout of the raises and ore chutes conforms to the contours of the ore body.

The application of the system may also prove necessary if the properties of the ground and ore correspond to those mentioned above, and in mining valuable ores where losses are to be kept down to the minimum.

## 9. Sublevel Caving

A general idea of this mining method may be gleaned from Fig. 369. A thick steeply dipping ore body is mined from top down by sublevels at an interval of 7-20 metres. The working sublevels are connected with the main haulageway by inclined rise headings (see Fig. 369). Inasmuch as foot wall rocks cave in completely at a given site of each sublevel after extraction, lateral raises have to be gradually carried up to secure connection with the air horizon, and from these raises short crosscuts must be driven to the ore body itself. In the ore body these crosscuts merge into cross drifts. The actual *stoping* of ore is done by a set of rather complex methods (described below).

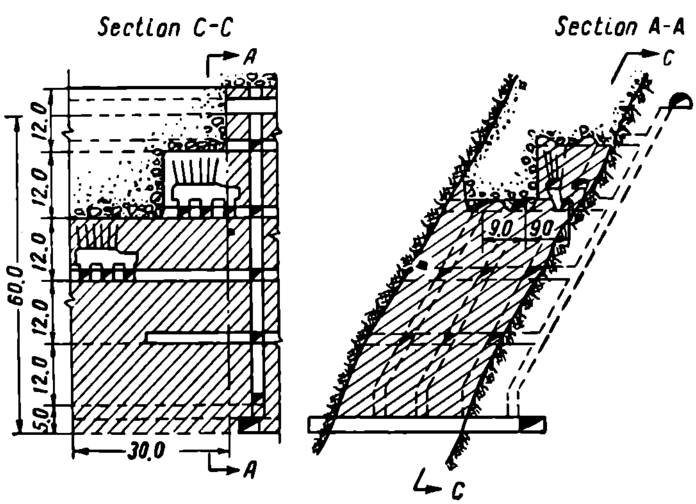



*Fig. 369.* Sublevel caving

Broken ore is hauled to discharge chutes, from whose gates it is loaded into mine cars in the haulage level. Mining is generally done from the hanging wall to the foot wall.

The sublevel caving system is very widely used in the Krivoi Rog iron ore district. With time the modes of stoping here have been modified to increase sublevel interval with the view to enhancing labour productivity and raising output in individual stopes, and to reduce at the same time the number of development openings in each sublevel. The sublevel interval has been increased by drilling longer holes with the aid of sectional steel. At first, the sublevel interval did not exceed 5-6 metres, while the actual extraction of ore was effected by inefficient side or slab cuts.

The first variation to supersede mining by slab cuts involved the use of "open rooms" (Fig. 370). Work in them, however, proved unsafe, and the method was therefore abandoned. As before, the sublevel interval in the open-room method remained at 5-6 metres and was later increased to 8-10 metres. The distance between the axes of the rooms was 7 metres. From the room ore was pulled out and hauled to an ore pass by a scraper. The trough-like shape of the room bottom made it possible for ore pieces to roll down to the scraper path. Apart from the above-mentioned danger, work in open rooms without mats entailed high ore losses and deterioration of its quality (30 per cent and over).

Another alternative for the extraction of ore, feasible also in working a weaker ore, became known as "pear-shaped side cuts or slabs" (Fig. 371). In this variant, small openings were cut at intervals of 7 metres on both sides of the crosscut or slice drift, and their

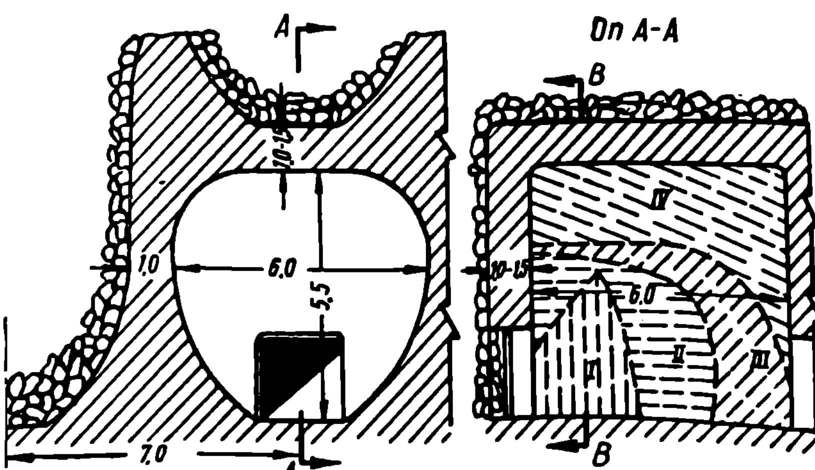



*Fig. 370.* "Open-room" mining

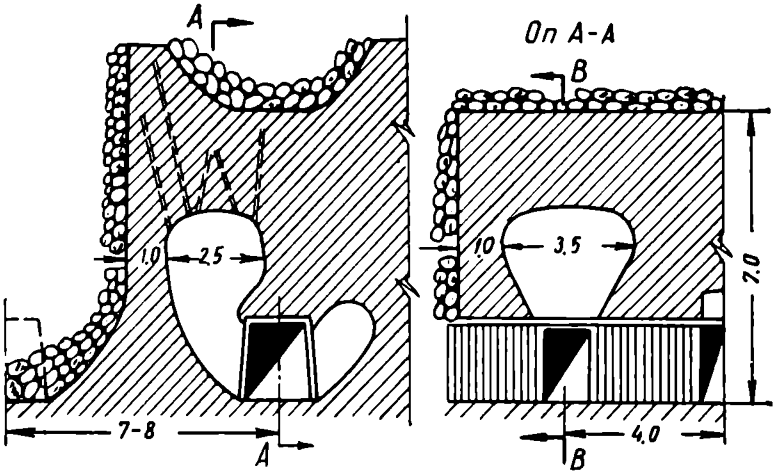



*Fig. 371.* Extraction by "pear-shaped" slabs

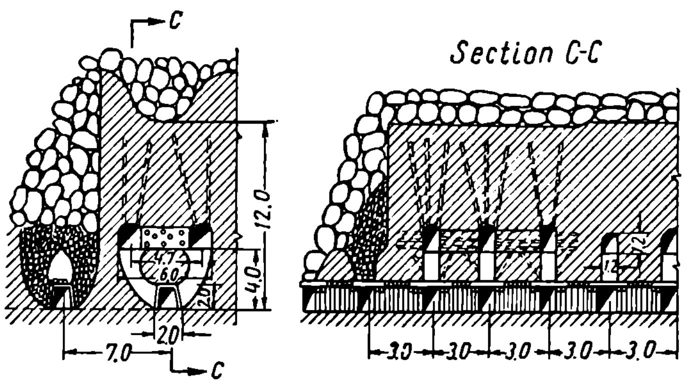



*Fig. 372.* "Closed fan" method of ore extraction

top portions were then enlarged to assume pear-like shape. The remaining ore was blasted by simultaneous firing of several holes. The method proved safer than that of "open rooms", but it did not eliminate large ore losses.

The modification of sublevel caving now employed in the Krivoi Rog district is one called locally "closed fan" (Fig. 372). It allows increasing the sublevel interval to 12 and more metres and involves driving up small inclined raises ("pull-holes") from the cross drifts, at intervals of 7 metres. They are subsequently connected with each other above these same cross drifts. From this break-through a series of up-holes 6-8 metres long are drilled. Their subsequent shooting

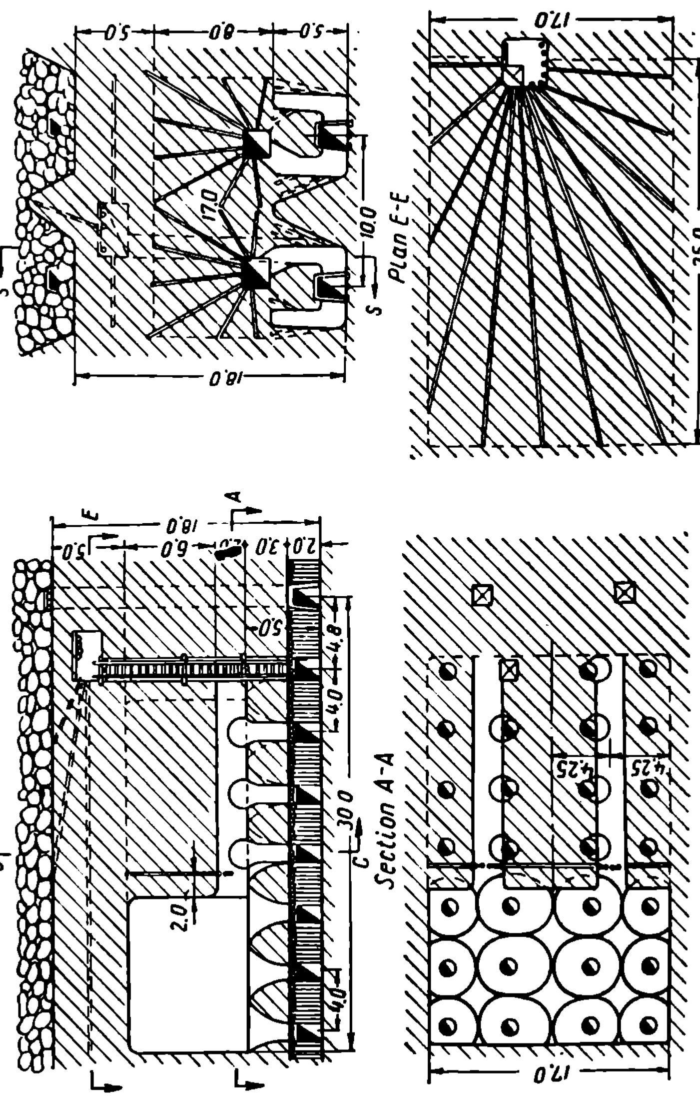



*Fig. 373.* **Long hole mining of ore**

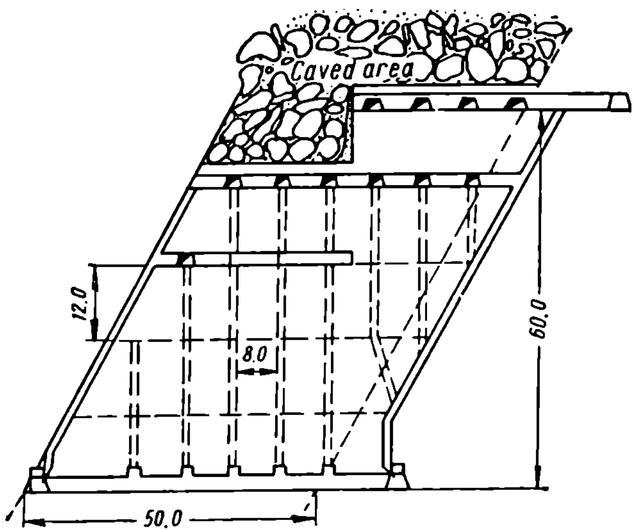



*Fig. 374.* Mining of ore by sublevel caving

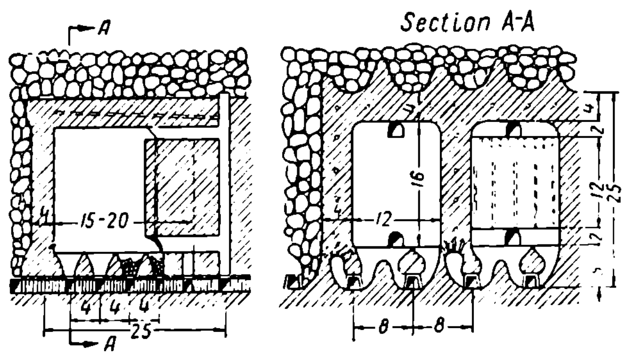



*Fig. 375.* A variant of stoping operations

breaks down the ore, which is then allowed to run into the cross drift. The ore is slushed to the ore pass. No mats are used in these operations.

Breaking of ore by deep blast-holes is being used more and more. One of the varieties of the method is depicted in Fig. 373. A block (locally "panel") 30 metres long, 17 metres wide and 18 metres high is mined at a time. The ore is drawn into *two* sublevel openings (drifts and crosscuts). The bottom portion of the block is worked by a method resembling the "closed fan", while the top is broken by horizontal or slightly inclined holes, which are also bored in fan-shaped rounds to facilitate the installation of drilling machines.

Sublevel caving, as it is practised today, is illustrated by Fig. 374. The level interval is about 60 metres and this is divided into sublevels 12-18 metres high.

The height can be increased still further if stoping is conducted in the manner shown in Fig. 375, which is called "stoping in rooms over pull-holes" in the Krivoi Rog iron ore district. The distinguishing traits of this alternative may be seen in the drawing.

The "closed fan" modification raises output per faceman to as much as 30 tons, while in long-hole blasting it is even 40-45 tons per shift.

Sublevel caving may be applied to recover the *rib* or *intermediary pillars* left between caved or worked-out and then filled rooms. This is illustrated by Fig. 376. From the lateral rise headings driven

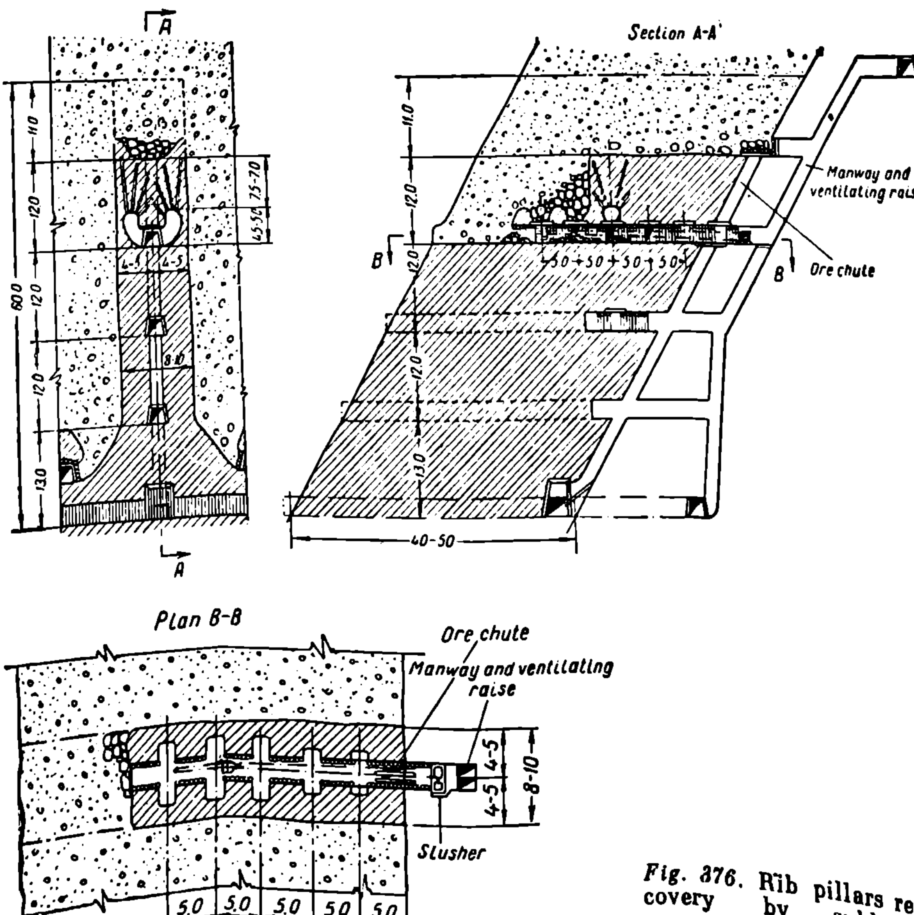



Fig. 376. Rib pillars recovery by sublevel caving

in the country rocks of the foot wall, short crosscuts are driven every 12 metres vertically opposite the centre of the rib pillar, and from these crosscuts cross drifts are run in the ore to the hanging wall. Ore is broken in the way depicted by Fig. 376. Blasted ore is pulled to discharge chutes by scrapers.

This mode of pillar robbing reduces labour efficiency, increases consumption of mine timber and makes ventilation of the stoping area difficult. Nevertheless, it makes it possible to cope with the difficult task of recovering ore from pillars left between the rooms worked-out earlier.

## 10. Methods of Mining with Induced Caving of Ore (Block Caving)

It has latterly become a common practice in Soviet metalliferous mines to use a method of mining involving mass caving of ore within a level (or sublevel) by blasting column charges of explosive. These are put into horizontally inclined or vertical blast-holes up to 40-45 metres deep, bored with the aid of special rotary drilling machines. This highly efficient method of underground breaking of ore originated in the Krivoi Rog iron district.

In this instance the natural process of spontaneous caving is replaced by one *induced* by blasting; hence the name of the system. The conduct of development work and stoping operations is illustrated by the typical example discussed below (Fig. 377).

A level with a 50-metre interval is prepared for stoping by lateral or fringe drifts made in the foot wall of the main and air horizons. To provide for a "loop" system of haulage a drift is run in the ore near the hanging wall and connected with a lateral drift by crosscuts and blind drifts (not shown in Fig. 377). The block of ore to be caved extends over 27 metres on strike and 42 metres across it. The pillar on the side facing a formerly worked-out block is 8 metres wide.

The development of the block starts with the driving up of air and service raises *1* outside the boundaries of the zone to be caved, near the hanging and foot walls. Carried parallel with this are ore passes *2*, at a distance of 11 metres between axes, along the strike. This is followed by driving cross *drifts 3 for scraping*, 8 metres above the haulage level, which are then connected by hanging and foot wall drifts of a relatively small section, driven in ore and serving as air and passageways to maintain the communication between scraper cross drifts.

The driving of crosscuts *3*, ore drifts *4* and the break-throughs connecting them with raises *1* opens up a wide front for the subsequent running of subordinate development workings, such as ore passes *5* and cross drifts on *discharge* or *draw hole level 6*.

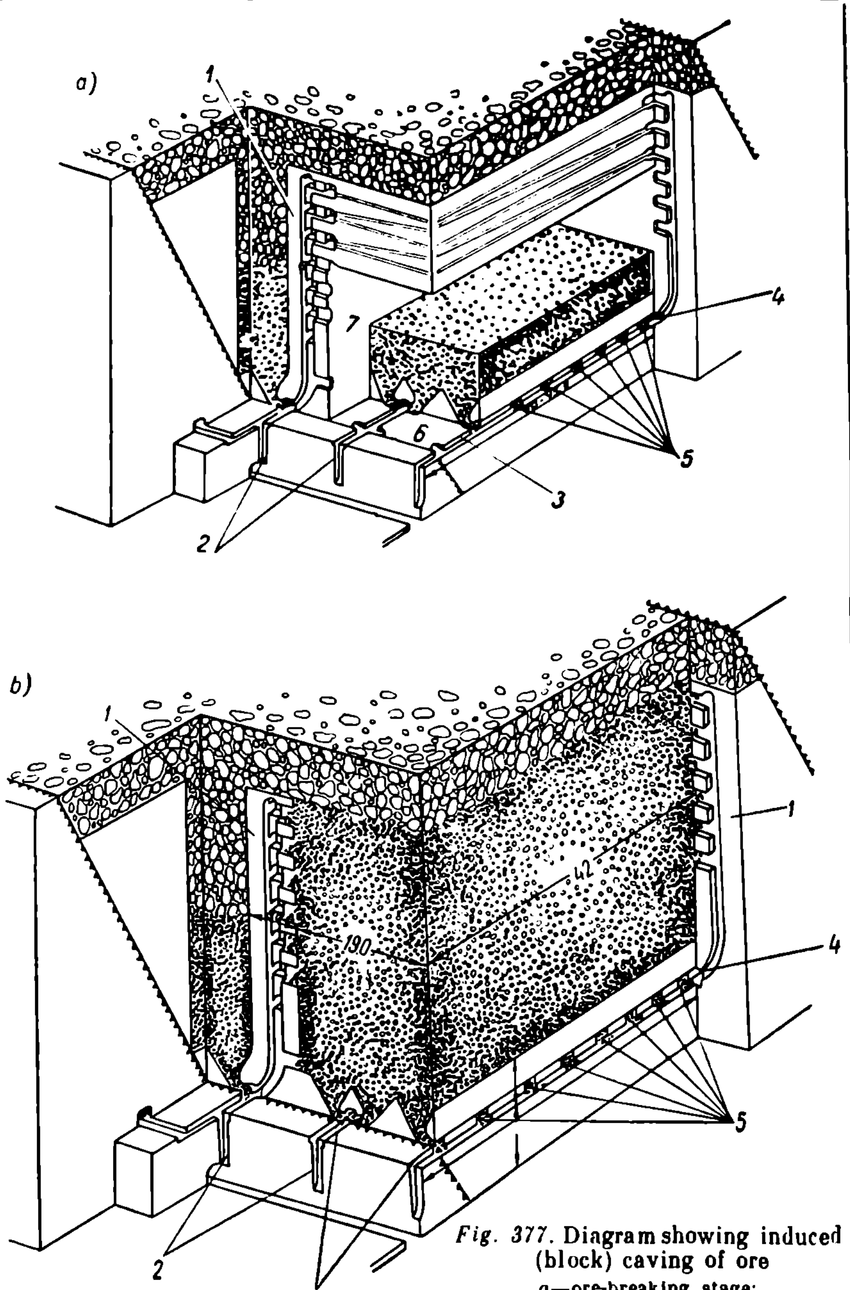



*Fig. 377.* Diagram showing induced (block) caving of ore

*a*—ore-breaking stage;
*b*—ore-drawing stage

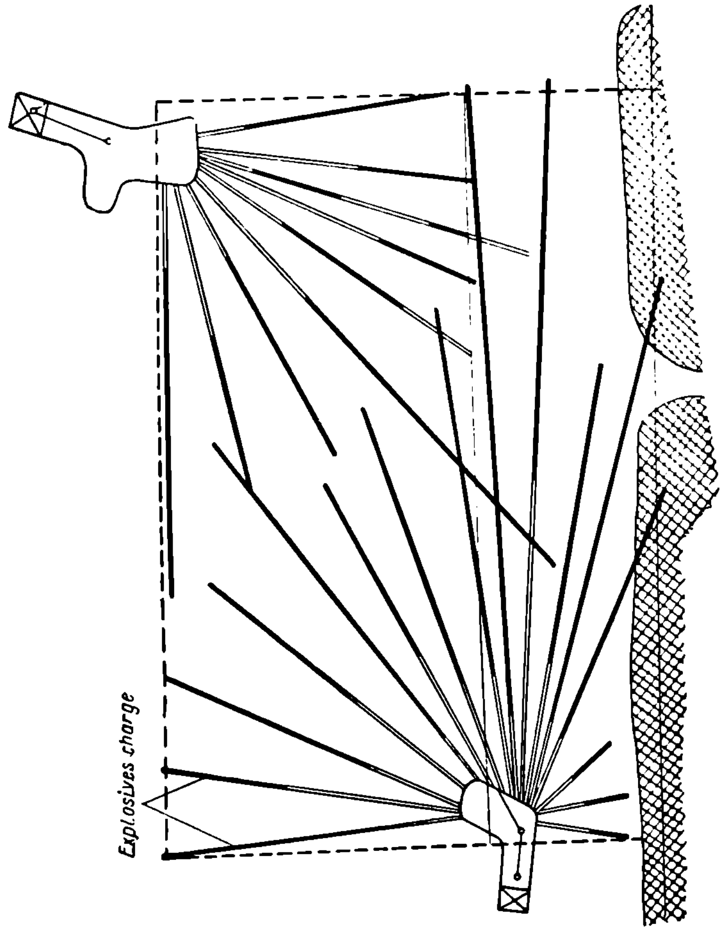



*Fig. 378.* Deep blast-hole round

Blast-holes are bored from drilling chambers (niches) arranged in raises *1*. These chambers accommodate drilling machines.

A small service hoist is installed in the air level to facilitate lowering and lifting heavy equipment. An electric power cable and water pipes are laid in raises *1* to the drilling machines. Each block has two to four machines to drill blast-holes.

Within the slice to be caved the blast-holes are arranged in fan-shaped rounds (Fig. 378), although the effect of blasting is somewhat better when they are parallel. The fan-shaped arrangement is superior to the parallel because it reduces the volume of labour-consuming development operations and the number of drilling-machine transfers, Krivoi Rog experience shows that the results of parallel and fan-shaped hole blasting are about the same.

The vertical spacing of blast-hole rows depends on the diameter of holes and the hardness of ore. Thus, in the case of ore of average hardness and blast-hole diameter of 100-110 mm the distance is about 4-4.5 metres, and with 80-90-mm holes—3-3.5 metres, depending on the strength of the ore.

The drilling of blast-holes is paralleled by the stoping of ore in *undercutting chamber 7* (Fig. 377), whose purpose it is to form an exposure surface for explosive action and to compensate for the loosening of ore in the block subject to caving. The volume of the undercutting chamber should come to about 30 per cent of that part of the solid ore mass which is to be blasted. If the ore in the undercutting chamber is insufficiently firm, temporary pillars are left, to be shot down first in the process of bulk blasting. Ore passes *5* are widened into hopper-shaped draw holes at the time when ore is drawn from the undercutting chamber. On the termination of these operations and after the thorough timbering of the mined-out scram level and the drilling of all blast-holes, they are charged with explosives. For this purpose use is made of special explosive sticks of an appropriate diameter and up to 0.5 metre long, or else ordinary sticks of ammonite or dinitrotoluene, bundled together into larger cartridges. These are delivered into the blast-hole by wooden charging sticks made of two-metre-long sections. A string (a strand of a hemp rope) or a piece of steel rope 6-8 mm thick is fastened to the first section of the charging stick. The explosive is charged in lots of two-three special cartridges (or in bunches of two-three ordinary sticks). The diameter of charging-stick sections is inferior to that of blast-holes. The string attached to the first section extends in the blast-hole on one side of the charging stick. After the delivery of the explosives into the hole, the charging stick is pulled out by this string.

The stemming is done with a mixture of clay and sand. When the rounds are fan-shaped, some of the blast-holes are filled with stemming to a considerable depth to avoid unnecessary breakage of ore in an area of their convergence near the drilling chamber (niche). The length of the stemming in such cases is determined graphically, the basic condition being the definite minimal distance between the charges.

Blast-hole charges are fired simultaneously within the boundaries of each slice with the aid of a detonating fuse. The charges in the slice itself are fired in rotation at intervals of 1 or 2 seconds. For slis purpose electric delay detonators are employed.

In the event a block to be caved borders on an earlier worked-out section, the support pillar left for the period needed to arrange an undercutting chamber is shot down together with the top portion of the block by firing the blast-holes drilled in it.

Fig. 379 is illustrative of a commutation pattern (connection diagram) in blasting the charges. The knife-switch closing the electric firing circuit is turned on on the surface, after the men had left the mine and measures had been taken to prevent an air blast (filling of draw holes with ore). Blasting operations are usually carried out on a day-off. The ventilation of the mine lasts 1-2 work shifts. The common practice is to call in a mine rescue station crew who take samples of the mine atmosphere by descending periodically into the pit with air-breathing masks. The samples are tested for CO, $NO_2$ and $CO_2$ content. Permission to resume work and lower men into the mine is given only when the level of obnoxious gases is again normal.

The *drawing off* of caved ore is started when the mine workings have been ventilated and put in order after the explosion. The broken ore (see Fig. 377*b*) discharged from ore transfer raises *5* on to the scram level is hauled by a scraper to ore pass *2*, loaded into

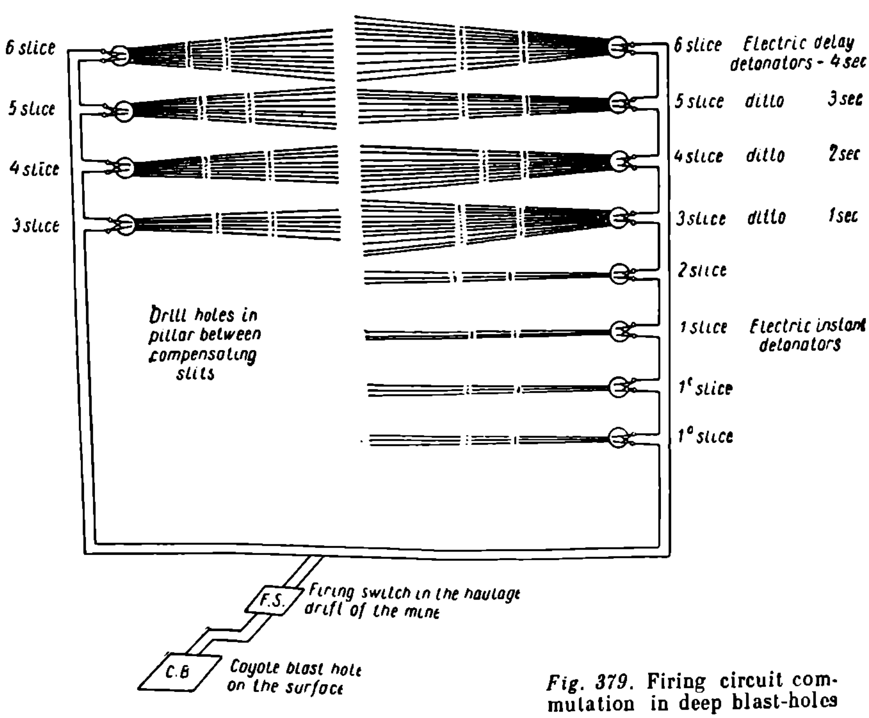



*Fig. 379.* Firing circuit commutation in deep blast-holes

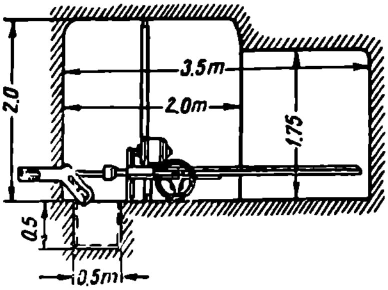


*Fig. 380.* A. Minyailo drilling machine setup

mine cars in the haulage leve and then taken in electric trains to the shaft.

In addition to a horizontal contact surface with barren cover rocks, the second and third series of extraction blocks have lateral ones, on the side of the sections earlier mined out and caved. Fig. 377*a* depicts an extraction block with one lateral contact surface, but there may be two and even three. To reduce ore losses in stoping the zone bordering on barren rocks, the number of lateral contact surfaces should not exceed two. In mining the extraction block shown in Fig. 377 ore from the draw holes under the pillar is last to be discharged. The ore pillars left at the foot and hanging walls are mined by other methods. Such *combined* extraction within one level is less economical because the increased number of its stages is likely to impair the efficacy of stoping.

Below are some details of the operations cited above. Blast-holes in the Krivoi Rog iron district are drilled by special machines designed by engineer A. Minyailo (Fig. 380), the Ore-Mining Research Institute and others. The drill bits are made at the mines of seamless steel pipes and tipped with tungsten carbide. Blast-holes are drilled at a rate of 2 to 20 and more metres per shift, this depending on the hardness and structure of ore and their diameter. Inasmuch as the increase in the length of a blast-hole requires a corresponding increase in time for inserting and withdrawing the drill steel, the efficiency of drilling somewhat drops.

The *drawing of ore* broken by bulk blasting is one of the most responsible operations determining the ultimate results of the method of mining with induced caving. In blocks with upright walls the most rational method is that of uniform consecutive drawing of equal portions of ore from all the draw holes in the bottom. In these conditions, the contact surface between the ore and the caved cover rocks subsides, but generally retains a horizontal position until it reaches a certain critical point, determined by the physical properties of ore, the diameter of the draw holes and their spacing. This method of ore drawing makes it possible to extract maximum pure ore before it becomes contaminated.

Successful discharge of ore requires an appropriate size of drawing chutes. There are instances in the United States of the discharge openings of drawing chutes measuring 1.2×3 metres. Their sides are made of reinforced concrete with steel plate stiffeners. These chutes have air-operated gates.

Studying the phenomena occurring in the drawing of ore, Prof. G. Malakhov has staged numerous experiments with special laboratory models. One such model was a box with a glass side wall and draw-hole-like openings in the bottom. The box was filled with coarse sand divided by horizontal bands of coloured charcoal powder to facilitate the observation of any shifts occurring in the sand layers during the modelling. Some of the tests aimed at drawing sand via one chute (Fig. 381). It may be seen that the mass of sand displaced during the drawing process assumes the shape of an ellipse, very much extended upward near the glass side wall of the model. Its spatial counterpart would obviously be a discharge *ellipsoid*. Consequently, to be drawn off from an entire block ore must be discharged simultaneously from a number of chutes. Fig. 382 shows the consecutive phases of such drawing.

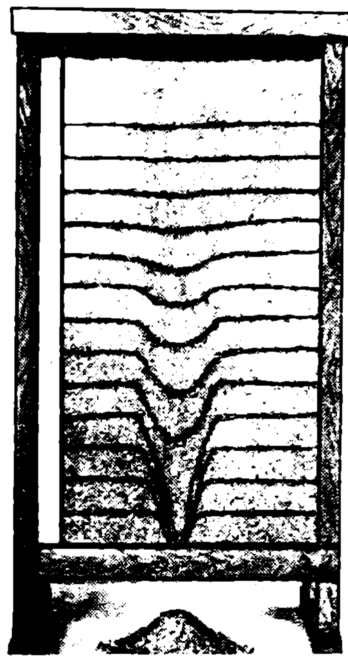


*Fig. 381.* Model testing of ore drawing through a single discharge chute (G. Malakhov's experiment)

As seen in Fig. 382, in the case of uniform drawing, the strip of charcoal originally (*a*) at the height of 30 cm subsides but retains an attitude close to a horizontal (*b*) over a distance of 17.5 cm. Below the critical height (12.5 cm), the contact surface acquires an undulating shape (*c*) and in the course of further drawing (*d*) changes into a series of funnel-shaped holes.

In pit conditions any further drawing of ore is accompanied by its dilution. In our mines ore is drawn consecutively uniformly, with the contact surface maintained horizontally.

Of much interest is the series of experiments conducted by Malakhov in drawing ore in conditions of high dip (Fig. 383). Here one sees the discharge ellipsoid grow in height vertically until it reaches the hanging wall, and then shifts to the rise. The ore is held up at the foot wall and so, when the dip is not too steep, additional workings with drawing holes have to be arranged in the foot wall at some distance between the haulage and air levels.

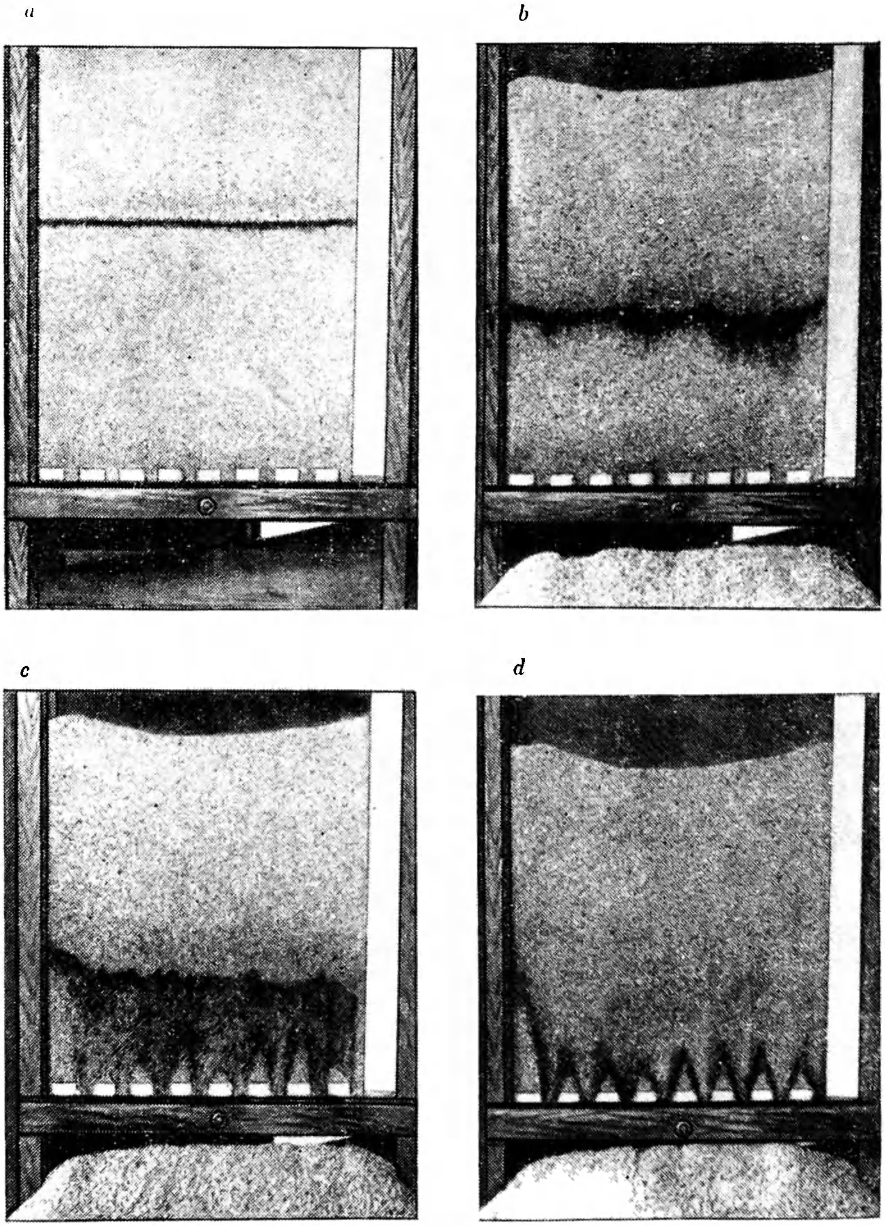



*Fig. 382.* Model testing of ore drawing through several discharge chutes

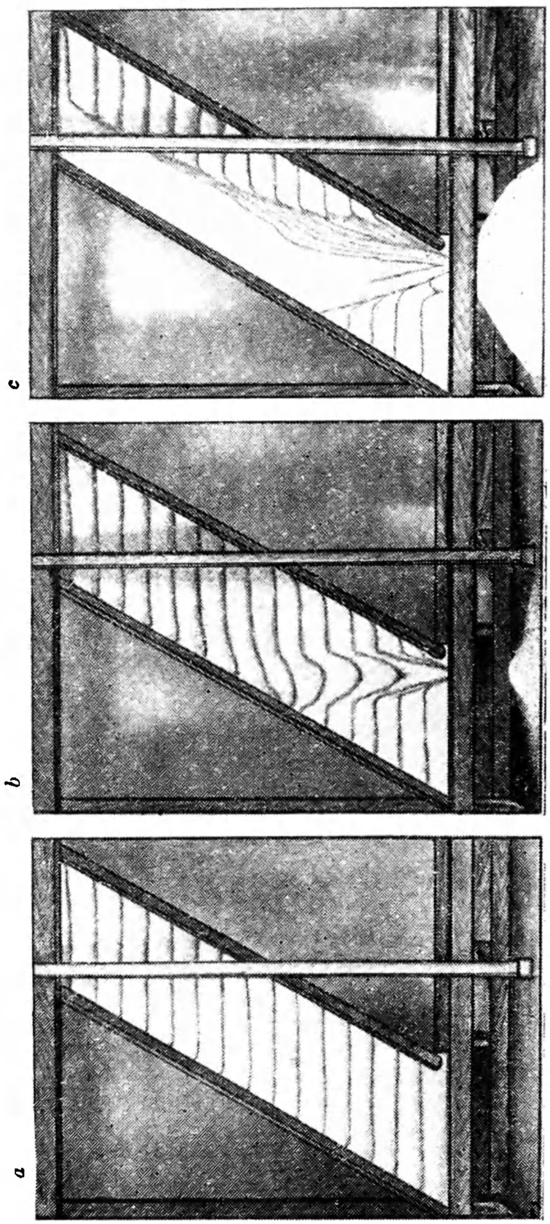



*Fig.* 383. Model testing of ore drawing in high dip

Before planning block caving operations, one must compile a special ore drawing chart (planogram).

When "chokes" are caused over a draw hole by ore hanging in large blocks, they are blasted out. The explosive charges, in sticks weighing 1-5 kg, are brought to the site of a choke on poles.

The variant of the induced caving (block-caving) system involving the arrangement of an undercutting chamber with an extensive exposure area, depicted in Fig. 377, is applicable in firm ore only.

If mined ore does not allow large exposure areas, the undercutting chamber should be replaced by vertical *compensation slots* with a total volume equal to that of the chamber but lesser exposure of the back (Fig. 384).

When the section of the deposit to be mined has been divided into a series of zones 20 metres wide on strike, lateral drifts *a* and air (and service) raises *b* are carried up. At the hanging wall these raises are run from the cross drifts of the secondary breaking level. From the raises, niches for drilling machines are cut out at definite intervals. Workings of secondary breaking level *c*, ore transfer raises *r* and crosscuts in draw level *o* are driven at the same time.

On the completion of development work and the drilling of blast-holes vertical compensation slots are formed in odd-numbered zones. Ore in these slots is broken by blast-holes or sectional steel long-holes from sublevel drifts. In the former case, the "catching" draw holes are widened over the entire area of the slot before the blast-holes are shot. In the latter case, the mining method is similar to that accepted in sublevel stoping.

During the caving of even-numbered zones it is the pillar between the compensation slots that is shot down first and then the overlying block of ore. After this, vertical compensation slots are formed in the even-numbered zones and these are also blasted. Ore is drawn from over the entire area of the caved extraction block. The latter's size differs from direction to direction. It is also possible to effect caving in only one zone.

Ore is discharged through draw holes on to grizzlies set up in bilateral blasting chambers and from there, after undergoing secondary breakage, it is loaded into mine cars in the main haulage level via ore chutes with gates. From each pair of branch raises (chutes) ore is drawn separately and as it flows by gravity slushing is not necessary.

The choice of the proper method of bottom section preparation (including the arrangement of a grizzly level or scram drifts) must take account of concrete mining conditions. To cut down the volume of development work and running maintenance costs, miners in the Krivoi Rog district usually give preference to scram levels.

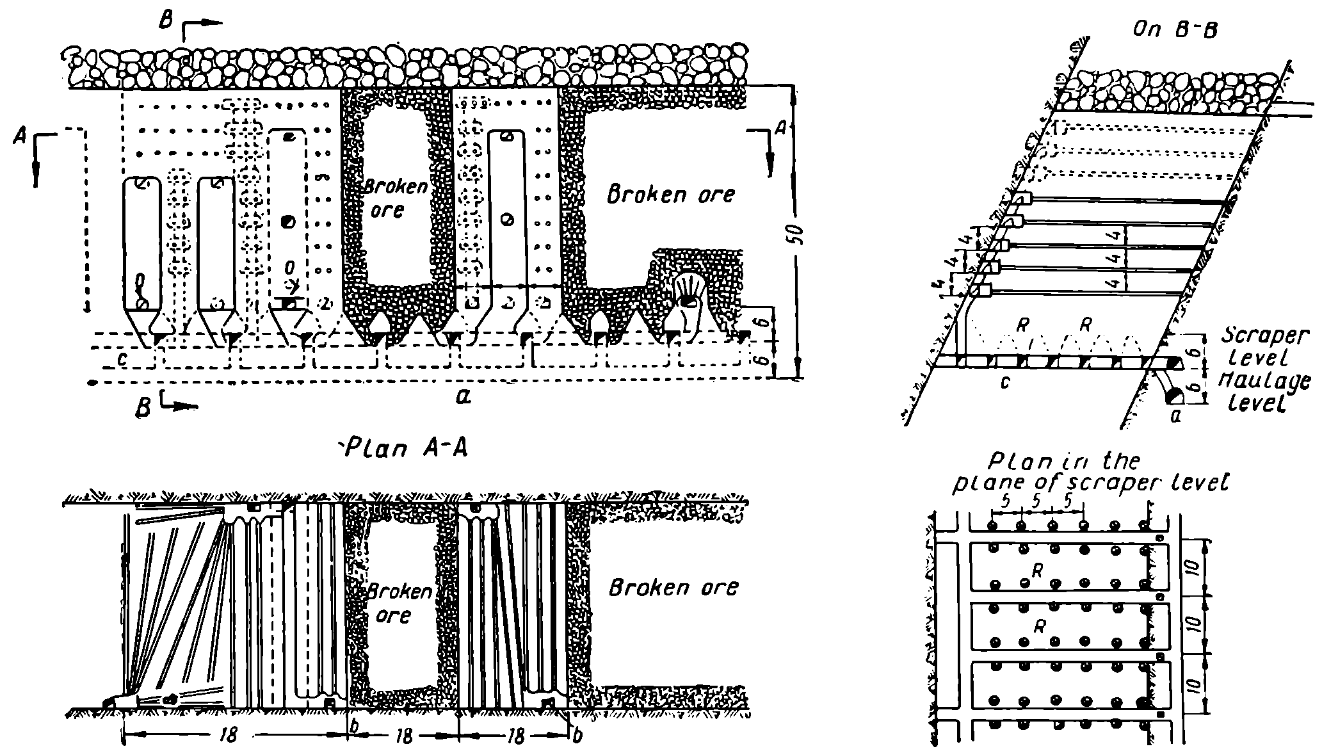



Fig. 384. Induced caving with compensation slots

Fig. 385 is illustrative of induced (block) caving in mining a flat dipping ore body and breaking ore by deep vertical blast-holes (up-holes). In this case, scram level 1 is arranged near the bottom of the ore deposit. Scram cross drifts 1 are run as shown in Fig. 385. On the sides of cross drifts niches are cut out and from there short inclined raises (branch raises) are driven, followed by draw holes receiving broken ore. Over the draw holes is the undercutting level. Undercutting covers the entire area of the solid mass of ore to be caved. To preclude premature downfalls, a portion of the ore block is temporarily left in the form of pillars.

To facilitate the drilling of blast-holes, special chambers *2* are excavated near the back of the deposit. Depending on the strength of the ore, they are 6 to 12 metres wide and 3.5-4 metres high. Cross-cuts *3* are run level with these chambers in the upper portion of temporary pillars to make it possible to drill blast-hole rounds in them. The latter are drilled from the chambers and cross drifts with shot-drilling machines, vertically downward, in rows spaced at $3 \times 3$ m both ways and with a diameter of 108 mm.

To obtain compensating space, only a part of the blast-holes are fired at first, and broken ore flows through draw holes to the scraper level. From there, via inclined ore passes (transfer raises) *4* driven in the rock of the foot wall, it is passed further on to the main haulage level. After sufficient compensating space has been obtained all the blast-holes are charged and fired and broken ore is discharged from all the draw holes.

The application o the induced (block) caving method allows an appreciable increase in efficiency and reduces the consumption of explosives. Mine timber consumption with this system is insignificant. All this considerably cuts the primary cost of one ton of ore stoped.

The percentage of ore dilution in induced caving amounts to 5-10, but when wall rocks include valuable components the recovery rate of available reserves may be as high as 100 per cent and over, with the metal content in drawn-off ore somewhat reduced.

The advantages of induced block caving are:

1) High degree of safety, since the miners engaged in stoping work in openings of small section (grizzly level, drilling chambers).

2) Reduced dust formation in the atmosphere of the mine, this being achieved through large-scale substitution of percussion for rotary drilling, which is of vast importance for combating silicosis. The secondary breaking (grizzly) level, however, continues to be a serious source of dust production.

3) High rate of production and labour efficiency, insignificant consumption of materials and low mining costs.

4) Abundance of broken ore in the mine, this making possible its uninterrupted drawing.

The disadvantages of the method:

1) Considerable time required for the development of extraction blocks.

2) Stringent regulations governing appropriate drawing of ore.

3) Large losses of ore and high rate of its dilution with waste as the result of unsuccessful blasts and inappropriate drawing of ore.

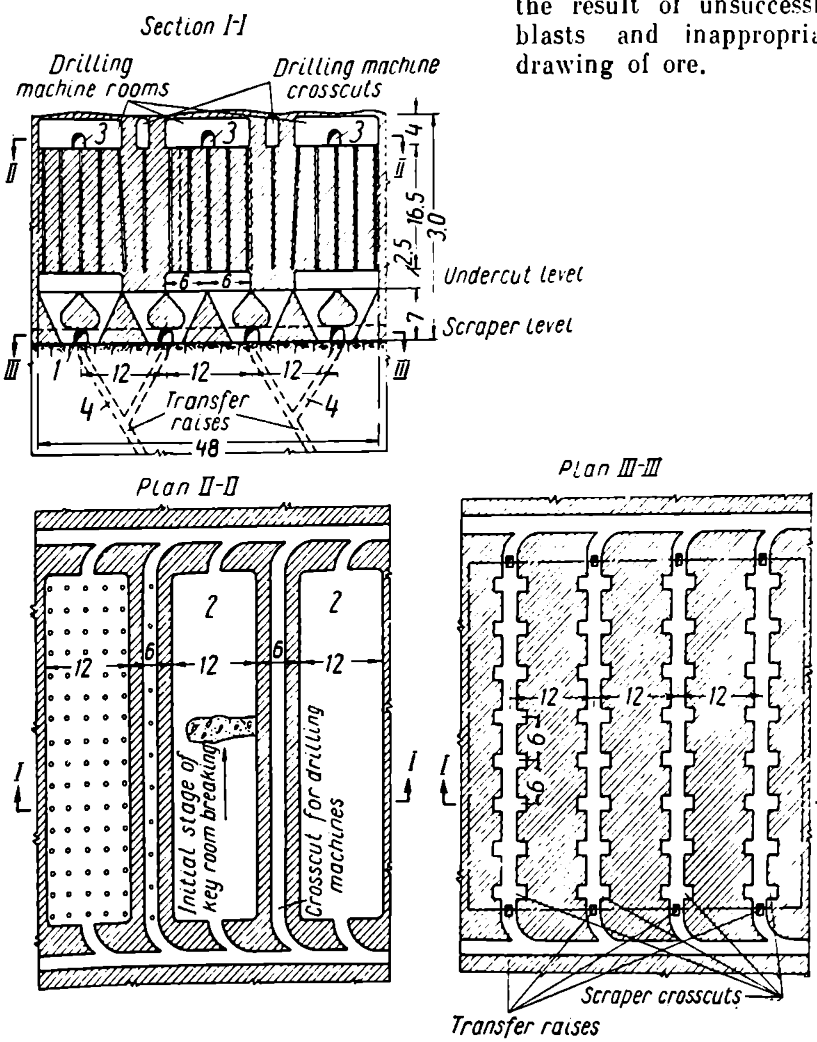



*Fig. 385.* Induced caving of ore and its breakage by the firing of charges in vertical blast-holes

## 11. Methods Involving Natural (Spontaneous) Caving of Ore

As in induced caving with this method of mining, the ore is preliminarily undercut, and after that it caves in under its *own weight.*

Fig. 386 depicts one of the current modifications of the system as applied in the Krivoi Rog iron district. The drawing shows an axonometrically projected portion of the extraction block in a very large deposit of weak ore (its coefficient of hardness, according to Protodyakonov, is from 3 to 6). The country rocks enclosing the ore body include hornfels, whose coefficient of hardness ranges from 6 to 8. They contain 35-37 per cent of iron. The extraction block is 60 metres high, 70 metres long and 50 metres wide.

The block is developed from main drift *2* and crosscuts *1*. At a height of 7 metres above the haulage level lies a *scram* (*scraper*) *horizon*, where drifts *4* and cross drifts *6* are driven every 10 metres.

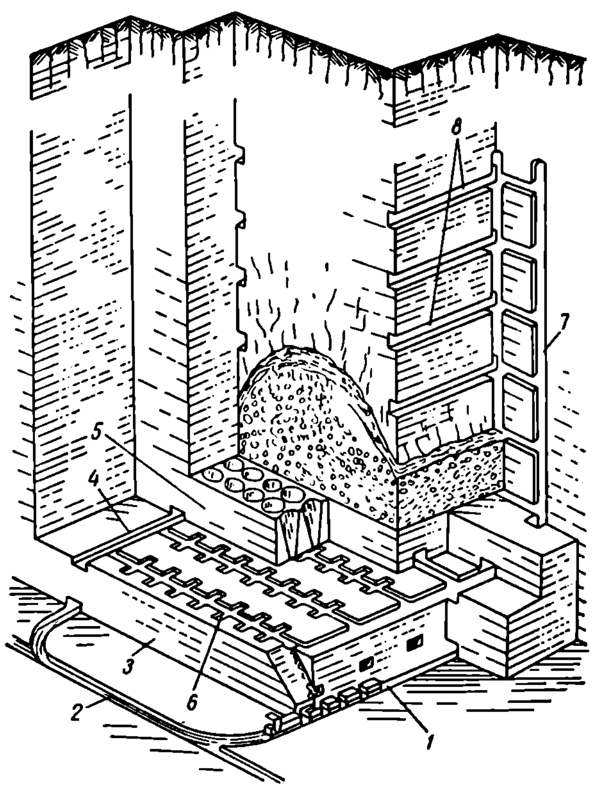



*Fig. 386.* Mining by spontaneous block caving

Ore mass *3*, seven metres thick and lying between the haulage and scraper levels, is called *scraper sublevel*. Lying over it is *draw hole sublevel 5*, which is 8 metres high. Ore discharged through draw holes (finger raises) is slushed to ore passes or transfer raises along the scraper level and is then chuted down to the haulage level to be loaded into mine cars.

The drifts and crosscuts in the scraper horizon are timbered with three-piece frame sets and the points of their interconnection are reinforced by metal square sets. To facilitate scraper travel, rails are laid on the floor of the scraper level drifts and the set posts are sheathed by an iron sheet at the bottom. The use of drag or plate conveyers in lieu of scrapers would reduce the section of haulage openings and that would increase their stability.

To weaken cohesion between the block to be caved and the solid mass of ore surrounding it, the block is preliminarily *cut off* by shrink drifts and crosscuts *8*. These openings are run every 8-10 metres from cut-out raise *7*.

Before it is caved, the block of ore is *undercut*. For that rounds of deep horizontal blast-holes are drilled over the draw-off raises in the *undercutting level*. To make possible their drilling and facilitate the breaking and fracturing of ore, a series of drifts and crosscuts are run in the undercutting level.

These are spaced so as to facilitate the complete destruction of ore pillars between the undercutting openings after the blast-holes have been fired.

As the blasted ore is drawn from the raises, the undercut block of ore spontaneously subsides and caves in under its own weight and that of the cover rocks (see Fig. 386). This is facilitated by the above-mentioned shrink or boundary drifts and crosscuts fringing the block. As it caves in and settles down, the ore fractures into finer pieces and flows by gravity via draw-off raises to the scraper level drifts, where scrapers haul it to transfer ore raises and pass it through chutes into the haulage level.

By registering the amount of ore drawn from each chute, it becomes possible to form a rough idea of the movement of ore occurring in the block when it caves in and settles down. Fig. 387, referring to the mining of the large Bolshevik iron ore deposit in the Krivoi Rog district, demonstrates the progress of caving and the position of the subsiding ore surface month by month in 1950. The first signs of ore dilution appeared after 2/3 of the ore reserves undercut for spontaneous caving had been drawn.

To prevent ore from being diluted by waste, it is drawn in accordance with an earlier compiled chart (planogram).

There are other variations of this method with no slushing operations, with ore being passed down to the haulage level through

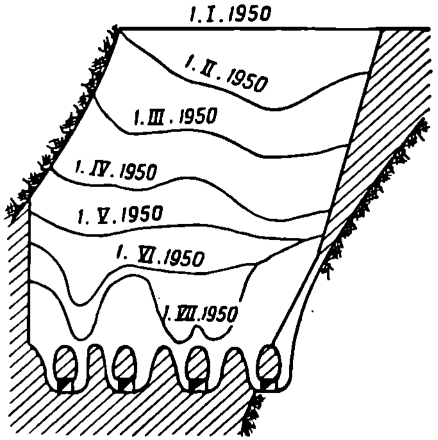



*Fig. 387.* Caving ore surface in different periods of its drawing

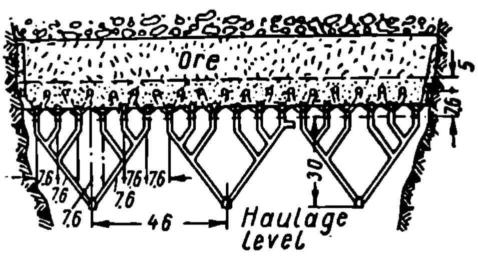



*Fig. 388.* Branch ore raise system

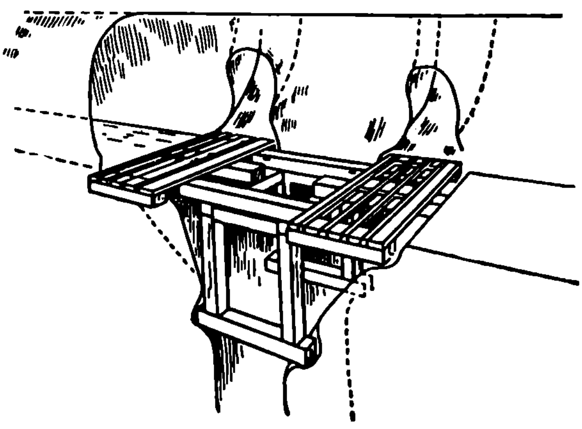


*Fig. 389.* Underground grizzlies

*branch (finger) raises* (Fig. 388), which we have already seen in Fig. 385. This modification, however, requires a great deal of complex development work in barren rocks.

If ore to be drawn contains large lumps, a *grizzly* or *secondary breaking level* is arranged below the draw-off or finger raises. The large pieces of ore may be broken in stationary grizzlies (Fig. 389) with hand tools or, if necessary, by small explosive charges.

Quite a number of conditions are necessary to ensure successful natural (spontaneous) block caving of ore. These are: 1) undercut ore should cave in and fracture into pieces making its drawing possible, and it must not be allowed to compact; 2) hanging wall rocks should be stronger than ore and apt to cave in in large blocks; 3) deposit must be large, not less than 20-25 metres thick with flat dip and not less than 25-30 metres in steep dip; 4) since the application of this method involves high ore losses (at least 20 per cent) and dilution with waste (to 30 and more per cent), it can be used only in mining poor ores; 5) the negative effect of dilution is somewhat attenuated when the enclosing country rocks contain ore matter (as is the case in the example above); 6) the elaboration of this constructively complex method of mining necessitates thorough preliminary exploration of the deposit.

If these rather rigid conditions are complied with, the system ensures safety of mining operations, high production and labour efficiency rates (60-80 tons per faceman per shift and 15-25 tons per underground worker per shift), reduced consumption of explosives and mine timber, low mining costs and the possibility of increasing the overall output of the mine as a whole.

## 12. Combined Methods of Controlling Enclosing Country Rocks

Underlying the classification of mining methods is the principle of controlling the enclosing country rocks. In certain mining systems these methods can be *combined* with each other.

Thus, quite distinct from other systems of mining is that of *square-set stoping*. At the same time, as stated above, the square sets quite frequently, though not always, are filled with *waste* to make them stronger. Equally, in singling out *stull-set* mining, we bear in mind that in most cases here the mined-out area is also supported by a *mass of mine-fill* or shrinkage-stoped ore.

In the working of thick deposits *shrinkage-stoped rooms* may, after ore has been drawn, be packed with *filling materials*, while rib pillars in the second phase of mining may be recovered by *horizontal slicing* or *sublevel caving*.

These examples are a good illustration of the fact that the choice of a proper system of mining does not necessarily imply adopting just one single method of country-rock control, for the combination of two and even several different methods may, depending on the conditions prevailing, prove quite justifiable.

## 13. Some Remarks on the Mining of Precious Stones (Gems)

Mining of the numerous occurrences of various gems is only in exceptional cases well-organised and technically equipped. One of the most important of these exceptions is the production of diamonds. Of other precious stones apart from diamonds, sapphire, amber and emerald are the only ones whose annual production comes to 1-2 million rubles. For the other gems the figure is below that. Hence, with the exception of the ones named above, the other precious stones are mined on a small scale. If all precious stones were classified into groups by geological origin and the proportion of each in the total value of world output were estimated in per cent, Academician A. Fersman says, 62.3 per cent of this value would be accounted by gems (almost exclusively diamonds) found in peridotite and basalt rocks and 35 per cent by those recovered in secondary bedding, among sands and talus. Only 2.7 per cent of the aggregate value of gem output represents the share of all the other groups, the most important being pegmatite veins in granites and granite contacts. This makes it quite clear that, barring diamonds, most gems, taking their value, are extracted from placers and the outcrops of pegmatite veins and, as already stated, in small-scale mining operations.

Extensive information on the original discoveries of precious stone occurrences, their search, prospecting and even mining may be found in Fersman's capital monograph *Precious and Semiprecious Stones in the U.S.S.R.*

In the Urals, gems are very often found when washing gold and platinum placers (for example, demantoid garnet in the Tagil district).

In the U.S.S.R. emerald occurrences are the only ones that are worked by technically equipped mining enterprises. Latterly, however, extremely rich *diamond* deposits have been discovered in Eastern Siberia and a diamond mining industry has now been set up.

Underground production of diamonds is so far biggest in South Africa. The diamond-bearing rocks (kimberlites—"blue ground" or "blue earth") occur in the shape of singular vertical cylindro-conical bodies (pipes). The average diamond content in this rock is 0.50-0.25 of a carat (0.2 g) per ton. The hardness of kimberlite is variable, but generally this rock is not too strong and weathers easily. The vertical spacing of levels is from 85 to 160 metres. The sublevel interval comes to 12 metres. The mining pattern (Fig. 390) is as follows. In the lower part of each sublevel a network of tramming openings driven at right angles to each other are arranged at intervals of 40-70 metres. These cut the sublevel into blocks with square bases. The workings have an arched ceiling with no support. Prior to extraction operations, each block is divided by subsidiary openings running perpendicularly to the boundaries of the deposit at 7-metre intervals. Developed "blue ground" blocks are mined by a *combined* method: partly by blasting in overhead stopes and partly by caving the mineral. In the latter case, it is shrinkage-stoped in production faces. Kimberlite in each sublevel is extracted from the boundary of the deposit towards the side nearest the hoisting shaft. Diamond-bearing rock is blasted and then shrinkage-stoped (see bottom sublevel in Fig. 390), with a portion of it, corresponding to the percentage expansion on loosening, being removed from the stopes. The latter may be left with almost no support. When these workings approach the top portion of the sublevel, part of the "blue ground" caves in under the weight of the overlying barren rocks. This is followed by the drawing of the shrinkage-stoped "blue ground", whose place is gradually taken up by the subsiding cover rocks.

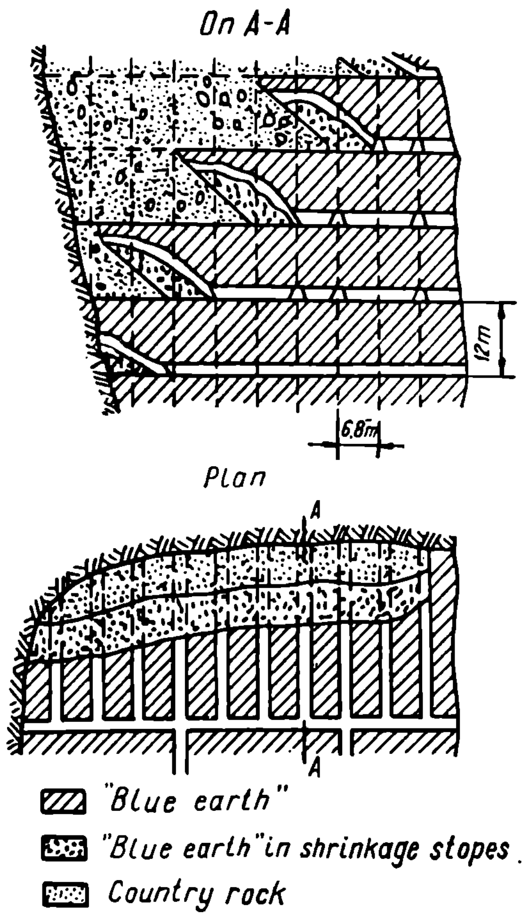



*Fig. 390.* Method of mining diamond bearing kimberlites

A certain amount of diamond-bearing rock in the top portion of the sublevel becomes mixed with waste and is lost. The caved ground lies at an angle of repose (around 40°). The latter stage of mining involves the overhand stoping of a new sublevel section, in the course of which the broken kimberlite falls on to the caved ground (Fig. 390).

These operations are carried out simultaneously in several sublevels. The mined diamond-bearing rock is then brought to round vertical transfer raises with a diameter of 1.8-3 metres and goes through them to the main haulage level.

The above described system may be included in the combined group, since the mineral is extracted partly in overhand stopes by the shrinkage method and partly made to cave in.

## 14. Mining by Glory Holes (Milling)

The idea lying back of this method implies driving a series of horizontal workings communicating with the surface via some permanent mine opening (shaft, adit, etc., Fig. 391). From these lateral workings vertical ore-pass raises, equipped with discharge chutes at the bottom, are carried up right to the surface. The mouths of these raises are made *funnel-shaped*, their ridges gradually extending to meet each other. The mineral blasted on the periphery of the funnels slides down the raises. Ore should be broken so as to avoid oversize pieces. Sometimes grizzlies made of solid poles are laid down at the bottom of mill holes to hold up large lumps and facilitate their breakage.

The method is a combination of open-cut and underground mining. Glory-hole mining largely depends on climatic conditions. Safety measures must be taken to prevent men working on the slopes of mill holes (safety belts, definite sequence of ore breaking, etc.) from falling. Sorting of ore and picking out of waste on the sides of mill holes are impossible. On the other hand, transportation of ore at the stopes is reduced to a minimum and there is no need of any kind of support. That makes it cheap to excavate ore.

Fig. 391 illustrates mining of an ore body with an irregular outline, dipping steeply, of an average thickness of 50 metres. The layout and size of glory holes are shown in the drawing. In this case broken ore goes to a grizzly level. Output per faceman per shift is 3 cu m, explosive and mine timber consumption is 0.8 kg/m$^3$ and 6 cu m per 1,000 cu m of ore, respectively. The underground workings are connected with the surface by an adit.

## 15. Basic Notions of Ore Leaching

It is common knowledge that mine water pumped from underground copper ore workings contains dissolved copper compounds. Iron objects submerged in this water acquire a coating of metallic copper. This method of obtaining copper is called *cementation*.

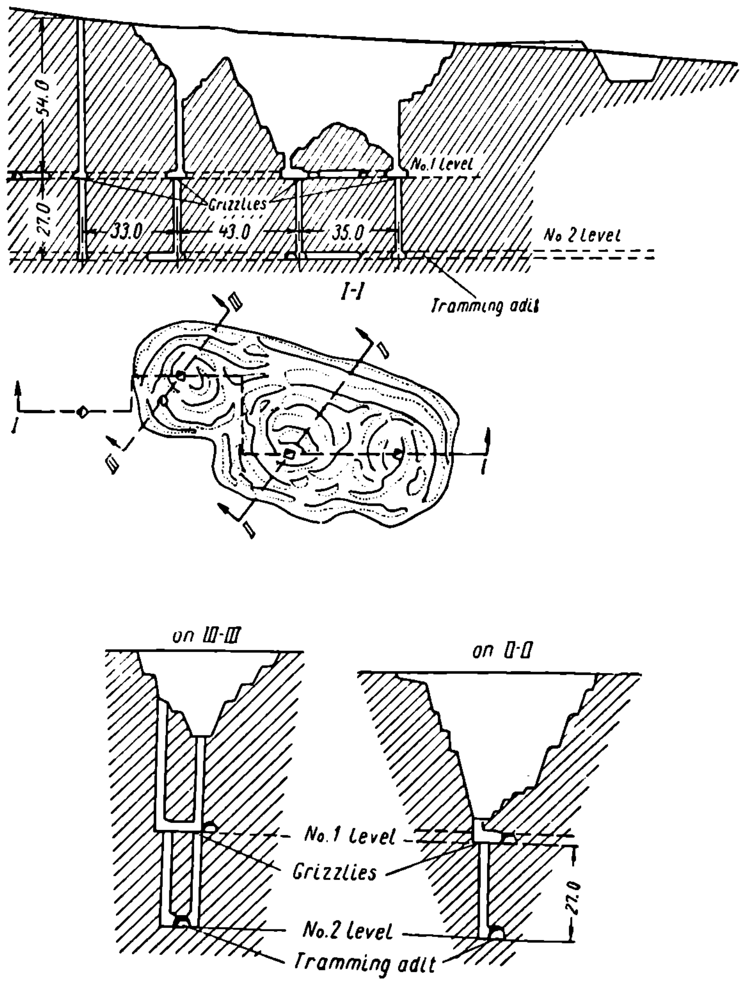



*Fig. 391.* Glory hole mining (milling)

This simple and cheap method of concomitant extraction of certain amounts of copper has been seized upon to obtain copper in old abandoned mines, whose mining entailed high losses of the mineral.

To extract large amounts of copper, one should not be satisfied with the natural flow of water into the mine, but get supplementary amounts of it to the old mined-out workings. To intensify leaching, a weak solution of sulphuric acid is sometimes pumped into a mine instead of water.

CHAPTER XXII

# MINING OF CONTIGUOUS BEDS

## 1. Undermining of Adjacent Beds

When beds, veins and ore bodies in general are close to each other, the excavation of one may sometimes unfavourably affect the extraction of the adjacent bed or ore body. This adverse effect on the conditions governing the mining of one bed, caused by the preliminary excavation of another, is termed *undermining* the first from below or from above.

Undermining has particularly dangerous consequences in the mining of coal deposits when, as is generally the case, a coal measure includes several and sometimes numerous working seams, whose extraction extends over large areas. Consequently, we shall devote our main attention to the possibilities of undermining and the measures to prevent this harmful practice and discuss in detail the problems pertaining to the mining of coal deposits, and shall confine ourselves but to occasional remarks concerning other minerals.

Working seams in a coal measure may lie at different distances from each other (normally to the bedding planes). If these distances are considerable, each seam can be worked by itself. If they are small, the working of one may badly affect the subsequent extraction of the adjacent seams. This may *undermine* the seams lying over the one to be extracted, and in certain conditions the movement of ground may also affect the *underlying* seams, and in this case it is usual to speak of excavating or "undermining" the seam from above.

Let us assume that over seam *a* at a distance *c*, there is seam *b* (Fig. 392, *I*). If underlying seam *a* is extracted first, the ground overlying it may be split by fissures. If the distance between the seams is insignificant, the fissures will reach the superjacent seam (Fig. 392, *II*) and make it difficult, if not totally impossible, to work it. In other words, seam *b* will be *undermined* from below.

If seam *a* is mined with complete filling, the caving and jointing of the ground over the worked-out area will manifest themselves insignificantly, and it will then be possible to excavate seam *b*. In this case there are good chances that, with complete filling, the cover rocks will not cave in and develop considerable fissures, and the overlying ground will come down smoothly without disrupting the continuity of seam *b* and complicating its extraction.

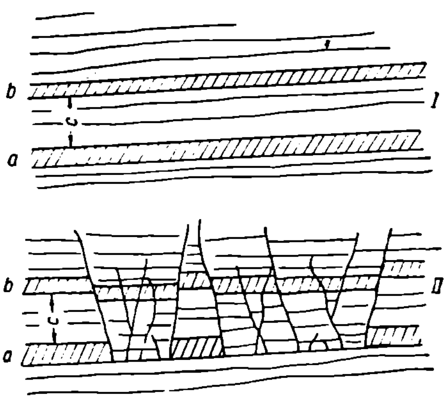



*Fig. 392.* Undermining a superjacent seam

Whether or not the seam is undermined depends not only on the thickness of the barren rock interspace between the seams and the method of controlling the underlying seam roof, but also on the thickness of the underlying seam itself, its angle of dip and the properties of country rocks.

Should local conditions make undermining possible, to prevent it stoping operations should be started in the overlying seams. This is important both for the safety in excavating the superjacent seams and for the preservation of the country's natural resources, since in undermined seams coal is lost irremediably.

Hence the mining of the underlying seam must lag behind that of the superjacent one. Let us determine the rate of this lag.

Fig. 393 depicts a vertical section on strike. Point *d* is the production face in seam *b* and point *f* that in seam *a*. The shortest critical distance between the faces—$x_{min}$, estimated along the strike, may be found provided the caving of the back down to the working face in the lower seam does not affect coal-drawing operations in the upper. To estimate this distance, line *ge* is drawn from the production face along the lower seam *a* at an expected angle $\delta$ of caving. For safety's sake, this angle in hard rocks should not exceed 75°, 50-55° in weak slates, and 30-35° in clays and sands. Point *e* must in all circumstances lie on the other side of the special face timbering in seam *b*, so as to prevent any possible collapse in seam *a* from involving the active stoping area in seam *b*. Inasmuch as the coordination of the positions and the advance rate of production faces in both seams may not be quite accurate it is necessary to leave a margin of about 15-20 metres between point *e* and the special timbering in seam *b*.

It should be emphasised that this may only help find the *minimal* advance distance of the faces. That may prove insufficient, however, for it is not only the top seam that can be undermined. Its extraction may adversely affect the conditions attending the working of the underlying seam. As a matter of fact, as seam *b* is mined, its cover

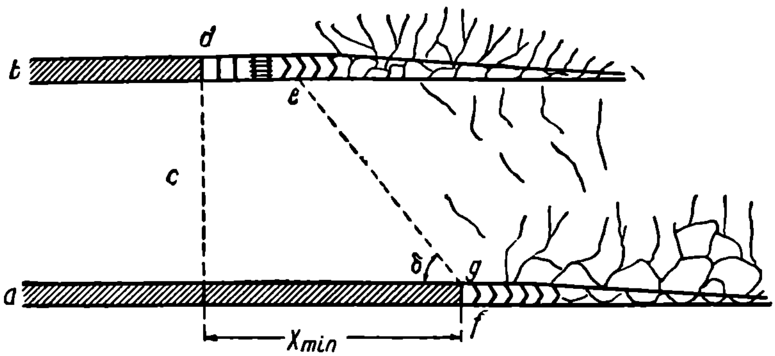



*Fig. 393.* Graphic determination of the advance rate in mining an overlying seam

rocks subside and settle down. These phenomena continue for a definite period, depending on the properties of the back, the thickness of the seam, the size and shape of worked-out areas and the method of roof control. In strong ground, the subsidence phase lasts longer than in plastic slates, which are liable to sag or warp. The settling rocks of the back in the upper seam may affect the interspace between the seams not only by static pressure, but also by dynamic impacts if settling and subsidence are brusque. With time these phenomena peter out and ultimately cease altogether, thus making it possible to start working the underlying seam without fearing any intense and unexpectedly growing pressure both from the rocks of the interspace and the back of the seam. The period in which one can safely proceed with the extraction of the underlying seam usually lasts several months. It may vary considerably because the thicker the interspace, the more complete is the recovery of the upper seam, that is, the smaller the coal losses during its mining and the weaker its cover rocks, the sooner one can start mining the underlying seam.

The after-effects of the working of an underlying seam on the mining of those above depends on the degree of recovery of coal reserves in the underlying seam, the size and shape of mined-out areas and the rate at which the production faces advance.

This may be explained as follows. The fuller (without coal losses) and quicker the recovery of coal in the seam over large areas, the sooner the capping rocks subside completely. Moreover, the ground in such instances comes down in one solid block, without causing any essential disruption of its continuity, and it is for this reason that the working seams which may also be involved in this stratum are not markedly disrupted, coming down gradually together with the huge massif of cover rocks. In this connection, the mining of seams by long continuous faces (walls), without the abandonment of any protective pillars in the worked-out area, is a factor which reduces the possibility of the superjacent seams being undermined.

Proceeding from a thorough study of conditions prevailing in the mines, A. Kilyachkov comes to the conclusion that in the cases of flat occurrence of seams in the Donets coal fields there is no cause to fear undermining if the thickness $M$ (in metres) of the interspace between the seams does not exceed the figure estimated according to the following formula:

$$M = 12\,m + 3.5\,m^2,$$

where $m$ is the working thickness of the lower seam in metres.

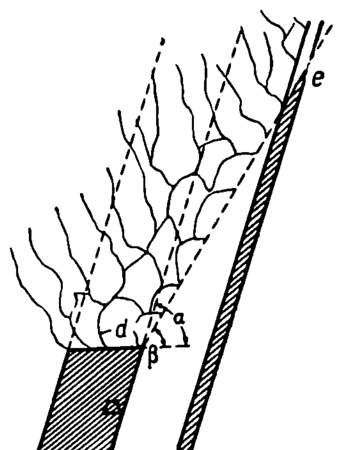



*Fig. 394.* Undermining from above a lower seam while extracting the one overlying it, in steep dip

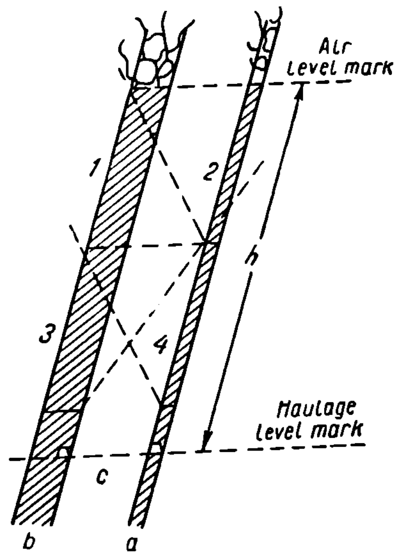



*Fig. 395.* Sequence of mining a series of contiguous steeply pitching seams

This formula applies to the usual complex of rocks encountered in the Donets coal fields—alternating clay and sandy shales—and also to instances of roof control in the underlying seam, both with partial filling in the form of waste pack walls and caving of the back.

In steep dip, the mining of a superjacent seam may possibly directly affect conditions attending the extraction of the underlying seam, that is, it may lead to the "undermining" from above of seam *a* by seam *b* (Fig. 394). This is possible when the angle of dip $\alpha$ is greater than angle $\beta$, along which the rock occurring at the bottom of seam *b* tends to slide down. If seam *a* lies close by, the boundary line of displaced ground *de* may cross interbedding space *c* and thus reach underlying seam *a*.

To forestall this danger, contiguous steep seams should be mined in the order indicated in Fig. 395. This drawing illustrates an instance when the floor is divided into two sublevels. It is clear that it is top sublevel *1* of seam *b* that has to be extracted first, since mining in upper sublevel *2* of lower seam *a* would result in the undermining of sublevel *1*. On the other hand, before proceeding to work lower sublevel *3* of seam *b* top sublevel *2* of seam *a* has to be preliminarily extracted, so as to prevent its being undermined from above. Hence the order to be followed in mining the sublevels in this case should be as indicated in Fig. 395. In the direction of strike the sublevel faces should be carried on with the corresponding rates of advance.

If the seams in a coal measure are contiguous, one should mine them in the order of extraction precluding their undermining from below or from above, and also bear in mind the layout of mine workings, some of which may be utilised in mining not just one but two or several seams simultaneously. Such workings may be lateral or inclined.

## 2. Examples Illustrating Mining of Contiguous Seams

There is at the Nesvetaiantratsit Trust Mine in the Donets coal fields a very flat-dipping seam called Nesvetaevsky which is 1.3-1.4 metres thick. Being separated by an interlayer, whose thickness gradually increases from a few centimetres to 6-7 metres, this seam—it extends over a considerable area—gradually splits into two independent seams: upper and lower. They are approximately equal in thickness, generally 0.5-0.7 metre.

So long as the thickness of the barren rock parting was below 1 metre, the two seams were mined as a single one, split into two individual benches, the waste from the parting being packed into mined-out space. But as the interbedding space became considerably thicker, simultaneous mining grew uneconomical and the seams were worked separately.

Originally the top seam was mined over extensive areas by ordinary machine walls and it was only later, after a few years (10-15), that the extraction of the underlying seam was started. It should be noted in this connection that this considerable time interval between the extraction of the seams was not due to any technical considerations but rather to unwillingness on the part of the mine administration to make any attempts to overcome the difficulties which were expected to arise in working the lower seam, separated from the old mined-out areas of the top seam by an interbedding only 1-6 metres thick.

When the drawing of coal was begun in the lower seam, it turned out that it could be mined by continuous faces (walls) up to 100 metres and more in length, with coal cutters and transportation of coal by low (since in some places the thickness of the seam dropped to 0.4 metre) flight-and-chain conveyers. It was also found that the longwalls could be supported by ordinary face timbering, involving the setting up of three-piece "frame" sets along with special support. Partial filling was built up of waste obtained by ripping the bottom of stone entries.

It is worthy of note that the heaviest pressure on both the development and production workings in the lower seam was recorded at the sites which lay under the coal pillars earlier left in the top seam, whereas in openings situated under the abandoned workings of the upper seam, where coal was recovered by longwalls, no particularly high pressure could be observed.

It goes without saying that, in conditions described above, the practice of drawing the lower seam many years after the extraction of the upper one by driving independent development openings in each one of the seams is wrong, since the main development openings in this case could be made common for both seams and the period between the mining of the upper and lower seams reduced to a few months.

2. In the Kuznetsk coal fields, two contiguous seams, one 1.4 and the other 2.2 metres thick, were worked at the Pioneer Mine. The seams were separated by an interbedding several metres thick and had both a flat and a heavy pitch.

In flat sections, each seam was worked by continuous faces (longwall variation). The production face in the top seam was carried on with an advance of 50-60 metres over that of the lower one. Both faces had ordinary timbering with double row of breaker posts for support. The main entries were maintained in the lower seam, while communication with the upper one was kept up via short slopes in the interbedding.

3. In conditions of steep dip (Fig. 396) the same seams were worked by the long-pillar method on strike with overhand stoping. The floor was divided into two sublevels, the working face in the upper seam being run 50-60 metres ahead. The extraction of the lower seam lagged about 100-200 metres behind the top seam. The main entries were maintained in the lower seam and communicated with the corresponding openings in the top seam through inclined break-throughs in the interbedding. Cribbing was employed to reinforce face timbering.

The above-cited examples show that mining of two or several contiguous seams must proceed according to plan with due account of the layout of mine workings in all of the seams and the sequence of their driving.

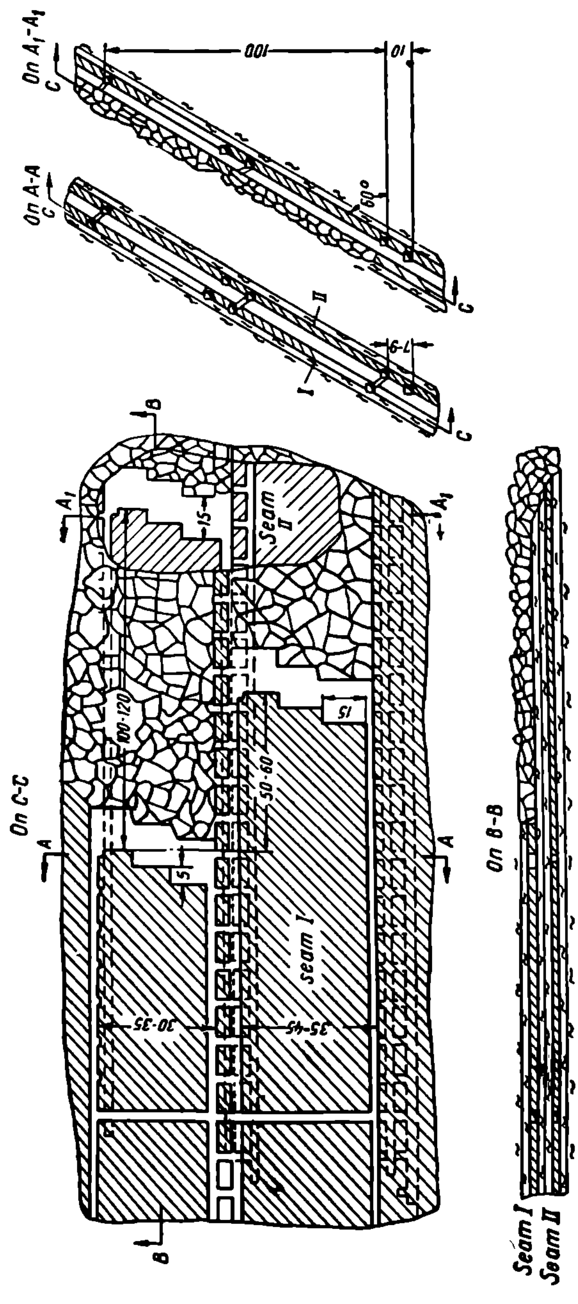



*Fig.* 396. Mining of two contiguous steeply pitching seams

## 3. Combined Development of Contiguous Seam Series

During the simultaneous mining of two or several contiguous seams, especially if they have a heavy pitch, it may be advisable to maintain entries not in all the working seams, but in one or a few. In such cases, the seams are connected by *crosscuts*, which are called *district* or *auxiliary* crosscuts to distinguish them from the main.

Fig. 397, for example, is illustrative of a case when, in mining two seams, $p_1$ and $p_2$, the haulage entry is maintained only in seam $p_2$. The seams are connected by auxiliary crosscuts *1*, *2*, *3*..., *n*, spaced at some distance from each other. The entry in seam $p_1$, running within the section between the main crosscut and the last auxiliary one, is *abandoned* (dash line in Fig. 397). Transportation of coal produced in both seams, air supply to the working faces and passage of men are all effected by the entry of seam $p_2$, which is therefore often referred to as *mother* entry. Since it serves a group of two or more seams it would be more apt to call it *group* entry.

In regard to production faces, the auxiliary or district crosscut may be *rear* (*back*) (*n* in Fig. 397), or *front* ($n+1$ in Fig. 397). Since in the process of driving front crosscuts the working sections of the entries run amid solid masses of coal their upkeep is easy. This also makes for better haulage conditions, eliminates air leakage and reduces coal losses in pillars.

The use of front crosscuts requires certain "overhauling" of loads during their transportation (see arrows in Fig. 397), but this drawback is insignificant compared to the above-cited advantages of the method, and for this reason it should be given preference.

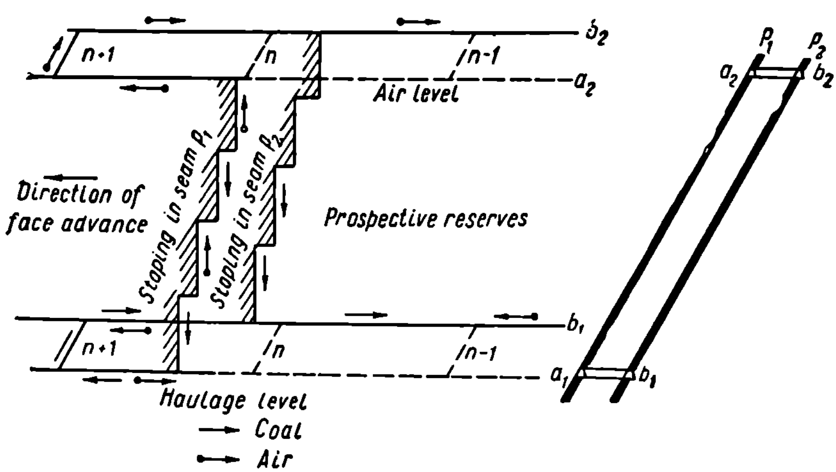



*Fig. 397.* Driving of a "group" (mother) entry in one of two working seams

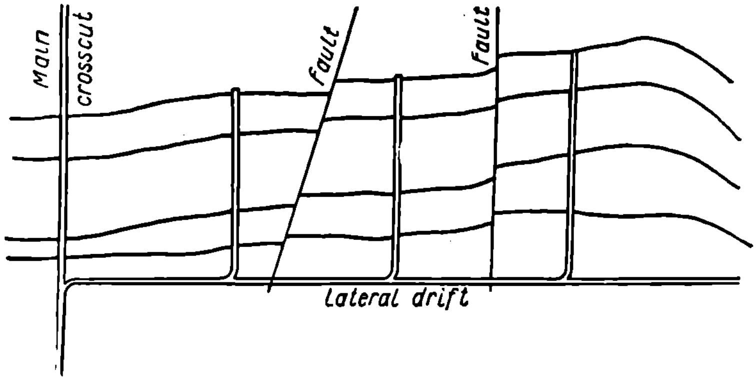



*Fig. 398.* Group lateral drift with irregular occurrence of seams

In mining steeply pitching seams, the maintenance of entries in a working seam is sometimes rather difficult, and even impossible in air levels. In such cases a *lateral* or *stone* drift may be run in country rocks or, to facilitate its driving, in a thin unproductive coal shed, if there is one available near the working seam. The lateral drift is connected with the working seam by auxiliary crosscuts.

If a group entry is driven and maintained in one of the working levels, it is desirable to have it closer to the foot wall of the seam series and its wall rocks stronger, so as to keep the costs of the entry's upkeep down to the minimum. Inasmuch there may be much traffic and air circulating in the group entries, their cross-section is made greater than in ordinary entries. Rock pressure in lateral drifts is not high and so they can be lined with concrete, which is also favourable from the point of view of reducing airflow resistance. The lateral drifts should be carried on rectilinearly over large distances and adapted to the general strike of the ground and not to the local irregularities in the working seams (see Fig. 398) so as to improve transport conditions. In the case of steep seams, the group or mother entry is used at first chiefly for the transportation of the mineral in the haulage level and then for the passage of air in the ventilating horizon. In other words, the service-life of group entries is usually twice as long as that of the level extraction period, that is, considerable.

Since making lateral or stone drifts is a costly affair, the need to have them should be substantiated technically (if, for instance, it should prove impossible to maintain airways in working seams) or justified economically. In medium and heavy pitch and in mining not-too-thin seams, it may happen that the lower haulage entries driven in working seams can stand quite well, whereas those running in the air level, undermined by the stoping operations in this very seam, may not only require extensive repairs, but even become totally impassable. Accordingly, there may be instances when entries are maintained in working seams in the lower level of a given floor, while in the air level lateral drifts have to be run in country rocks.

The spacing between auxiliary crosscuts depends on the difference between the cost of their driving and that of maintaining haulage entries and transporting the mineral along them. Analytical determination of these distances may be effected by the method described in Section 4, Chapter XIII for estimating the length of working sections. These distances are usually of the order of several hundred metres.

### 4. Significance of Group Openings in Mining of Self-Igniting Seams

As stated earlier, the driving of auxiliary (district) crosscuts from group or mother entries to working seams cuts down the expenses of maintaining openings and haulage tracks. In addition to these advantages, auxiliary crosscuts are also of considerable importance in working seams containing *self-igniting* coal, since they make it possible to seal off sections already stricken by fire or those exposed to the danger of its spreading from other parts of the mine. For instance, if a fire has broken out on account of spontaneous combustion somewhere between the production faces and a crosscut in seam $p_1$ (see Fig. 397), it suffices to provide *airtight fire bulkheads* in the crosscuts of haulage and air levels in order to seal it off. It is assumed that in these circumstances a coal pillar extending up the level interval is left between the above-cited crosscuts. If a given section is successfully extracted, this coal pillar can be partially recovered, or rather as far as it is technically possible.

Underground fires due to self-ignition sometimes spring up in goafs containing some amounts of abandoned coal, and this a long time after the completion of actual mining. For this reason it should be the rule to *seal off* all mined-out sections and panels *by bulkheads* even if there have never been any signs of fire recorded in them.

The best *preventive* measure against oxidation and heating of coal, which are liable to cause a fire, is to stop the access of outside air to the worked-out space.

If the seams within the bounds of panels (uni- or bilateral) are mined in the *retreating order*, that is, towards the district crosscut, the fire bulkheads may be put up in the entries proper instead, this permitting to reduce coal losses in the burning sections.

To *extinguish* more quickly flames that have been sealed off by fire bulkheads, mine workings and open goafs may be partially silted through special boreholes. Sometimes, when no outbreak of fire has

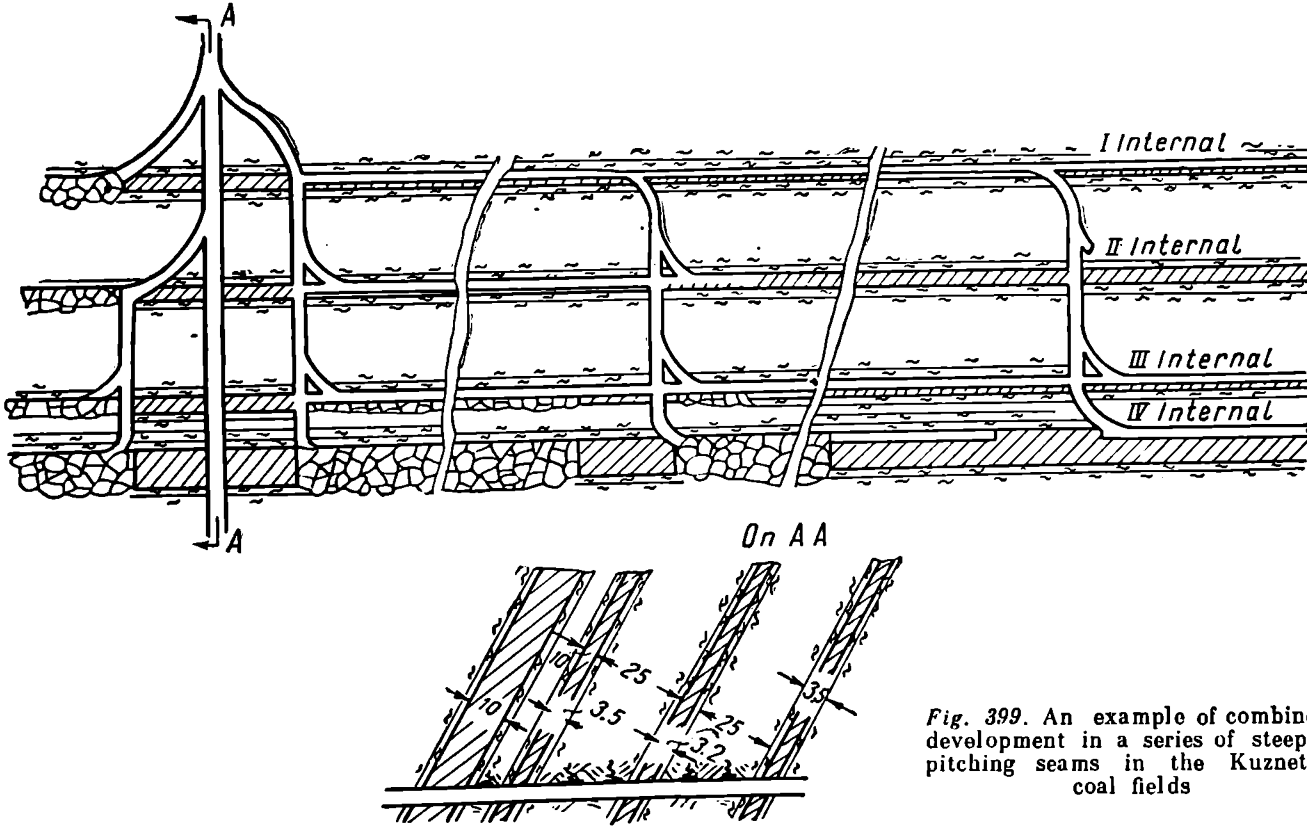



*Fig. 399.* An example of combined development in a series of steeply pitching seams in the Kuznetsk coal fields

been established, but the properties of coal do not exclude it, mine sections isolated by fire seals are silted as a precaution.

Group entries are widely used in the Prokopyevsk-Kiselyovsk district of the Kuznetsk coal fields where, as stated above, rich heavily pitching coal measures with self-igniting coal are mined. The development of a series of contiguous seams in this district is depicted in Fig. 399. The thickness of seams Internal *IV*, *III*, *II* and *I*, starting with the top, and the distances separating them are given in the drawing. The Internal *IV* seam is worked by the shield mining method; the others by pillar mining on strike. From the main crosscut a group entry is driven in the bottom Internal *I* seam. From this auxiliary crosscuts are run every 250-350 metres and from them entries to all the working seams. The panels in the mine field are worked from the shaft to the mine field boundary. The seams in a panel are mined simultaneously, but with the faces of the overlying ones carried on somewhat ahead. These seams are usually extracted by the retreat system.

## 5. Basic Concept of Combined Development of Seam Series by Blind Shafts

The Ruhr coal fields in West Germany have long been using the system of developing series of inclined and heavily pitching seams by blind shafts, graphically illustrated by Fig. 400. In principle the method consists in raising a vertical shaft, equipped with a hoisting plant, or a gravity incline (that is, an opening furnished with a plant for lowering coal in mine cars by their own weight) up the entire level interval, extracted by subfloors of insignificant height, from which crosscuts are driven to each sublevel. These blind shafts and groups of crosscuts are several hundred metres apart. The large volumes of waste obtained in sinking shafts and driving crosscuts are utilised for filling.

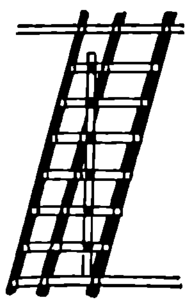


*Fig. 400.* Development of a series of seams by blind shafts

Investigations into the possibility of using this method in the Donets coal field have led to a negative conclusion in view of present mining trends in general and the tendency to employ large-capacity mine cars.

CHAPTER XXIII

# EFFECTS OF UNDERGROUND EXCAVATIONS ON THE GROUND SURFACE

## 1. Manifestations of Rock Movements on the Surface

Chapter VII showed that underground mining with *support pillars* of adequately calculated size did not affect in any practical way the overlying ground and, consequently, the surface of the earth's crust.

We have also seen that in working deposits with *caving* there may, generally speaking, appear three distinct *zones* over the mined-out area: that of *caving*, *sagging* or *subsidence* with fractures, and smooth or gradual *sagging* (Fig. 401).

It is to be noted that if the area of mined-out space is insignificant, the movement of ground over it may stop completely at a certain elevation.

Manifestations of ground movement *on the surface* may be widely different in nature.

Depending on the depth at which deposit is worked and the height of the above-cited zones, the surface area may find itself in the caving zone (Fig. 401, *III*), the sagging zone with fractures (*II*), or higher up, in the zone (*I*) distinguished by gradual sagging or, possibly, by complete absence of any ground movement.

The part of the surface involved in the ground movement is called *draw*. If the draw lies in the zone of sagging, it is limited by the gradual subsidence of the surface at its edges. But when the draw is in the zone of caving, rupture of the ground with the attendant fractures and even sinkings occur at its periphery along with the sagging.

In the first instance, the buildings and other structures on the surface will not be damaged. In the second case, the deposit occurs nearer the surface, and there is then a possibility of surface facilities sagging and leaning over. In the third case, when the surface lies in the zone of caving, it may develop fractures and sinks or pits, which would make it impossible to erect any structures in the affected area.

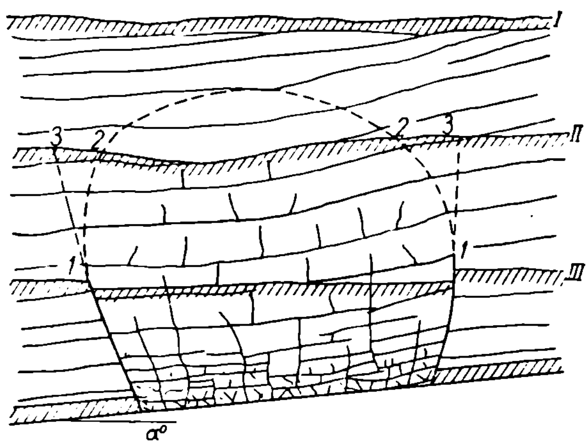



*Fig. 401.* Ground surface with respect to three zones: that of caving, subsidence and gradual sagging

Consequently, underground excavations can affect the surface in a variety of ways. The latter do not depend only on the depth of excavations, but also on the following factors: 1) thickness and number of working seams; 2) filling, its properties and the degree of completeness; 3) properties of the ground overlying the deposit; 4) angle of dip of the deposit; 5) shape and size of worked-out areas; 6) percentage recovery of the valuable mineral; 7) advance rate of production faces; 8) time elapsed since the extraction of the mineral.

When large deposits are extracted, the original attitude of enclosing country rocks, all other conditions being equal, is subject to greater disturbances. The filling of mined-out space does not completely eliminate the movement of ground; it still manifests itself, though in a much moderate manner. Because of their physical and mechanical properties, caving and settling rocks behave by far not in the same way. The stronger rocks rupture brusquely and subside in large masses (for instance, sandstone, limestone, most of the metamorphic and igneous rocks). Other rocks, in subsiding, sag gradually, developing fewer gross fractures and crevices. The more plastic are clay shales and clays. Noncemented sands cave in as does typical loose ground. Quicksands (that is, water-saturated silty ground) when involved in roof caving may cause inrushes by flowing out or erupting as liquid or viscous masses.

These properties obviously tend to affect the nature of rock movement over the worked-out areas and, consequently, the dislocation of ground on the surface.

The more extensive the mined-out area in a deposit, the greater the total front of working faces and the higher the degree of recovery,

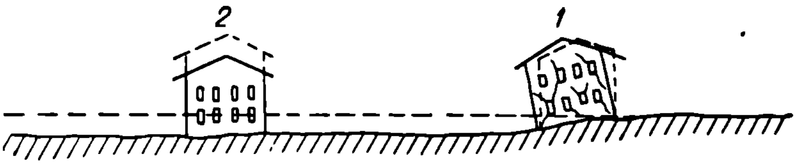



*Fig. 402.* Surface structures in the subsidence zone

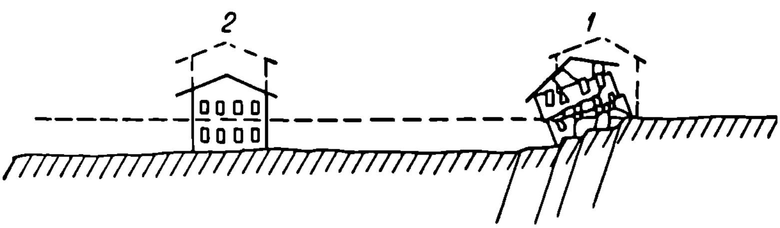



*Fig. 403.* Surface structures in the caving zone

the faster and more uniform is the movement of the ground, the sooner it ends and the more uniform is the subsidence of the surface. The uniformity of surface movement is also due to the more rapid advance of production faces. Generally speaking, all the manifestations of ground movement, occurring parallel with the advancing extraction of the deposit, should be regarded from a *dynamic* angle as a complex picture changing in space and time.

A structure in a zone affected by sagging (Fig. 402) may find itself on its edge (*1*) or nearer the centre (*2*). An analogous situation is encountered when the surface is within the caving zone (Fig. 403). The more dangerous is obviously situation *1*, where the structure lies at the edge of the subsidence or caving draw affected by depressions (saggings) or violent subsidence of the ground along the fracture lines. In this connection it should be noted that the edges of depressions and caving draws are liable to shift when the mined-out area beneath the draw expands with the advance of working faces. In such cases, sites *1*, which are dangerous for surface structures, are apt to shift and may eventually appear under various buildings and plants and destroy them.

The ground movement may cause vertical and lateral dislocations, and sometimes even uplifts or rises.

Lateral movements of the surface occur during the sagging and rupture of the ground and the formation of fractures, as well as following the compression and crushing of solid ground edges which are thus displaced towards cauldrons and caving draws. Paradoxical as it may seem at first glance, the phenomenon of surface rises following the movement of ground is explained by the fact that lateral dislocations of rocks may lead to their compression and extrusion in upward direction. For example, when the large Verkhnaya Marianna seam in the Karaganda coal fields was mined out by rooms close to the surface, there were extensive rock slides over the worked-out areas entailing ground extrusions in the shape of peaked ridges about 0.5 metre high.

Lateral displacements are dangerous for surface structures because the tensions they cause may be accompanied by ruptures and ultimate destruction of buildings.

Deformation of the surface is liable to manifest itself particularly violently in places where large shallow-seated deposits are worked by the caving method. One example is the Prokopyevsk-Kiselyovsk district in the Kuznetsk coal fields, often cited in this book, where rich heavily pitching coal measures are mined at a small depth from the surface. Here deep and extensive sinks and pits appear on the surface, generally running along the strike of the seams.

Huge sinks also appear in mining ore bodies by the caving of cover rocks, especially in top slicing and block caving.

Rocks caving over mined-out areas become loose and thus expand somewhat in total volume, which may diminish again later on, under the effect of their weight (so-called contraction or compaction of rocks). Rocks with different properties behave differently in this respect. Loose rocks, when they cave, remain practically unchanged in volume. Argillaceous rocks, being plastic, compact well to near their original volume. For such rocks the subsidence zone with a large mined-out area may be very extensive. And quite the contrary, strong rocks, which break into large blocks in the process of caving, contract insignificantly.

The subsequent decrease in the volume of displaced ground sometimes lasts very long—months and even years.

## 2. Safety Pillars

To avoid the destruction of underground mine workings or surface structures in the zones of subsidence (sagging) and caving, special *safety pillars*, that is, intact solid masses of useful mineral, may be left behind.

Among the underground openings of particular importance is the protection of hoisting shafts from movements of the ground for even their slight distortion creates considerable inconveniences, may prove dangerous for the operation of the hoisting plant and usually damages the shaft support. In addition to this, it is at the mouths of the hoist shafts that large surface structures and buildings—shaft houses, head frames, hoist engine and change-and-office houses, storage and discharge hoppers, concentration and dressing mills, etc.—are situated. For this reason it is the shafts and the surface buildings and the plants adjoining them that must first be protected from the ground movement by safety pillars. There may also be other surface structures that have to be protected from these hazards (see below).

In order adequately to plan the layout and size of safety pillars, one must know precisely the disposition of surfaces delimiting the subsidence and caving zones appearing over the worked-out areas in different conditions of deposit mining.

In the U.S.S.R. investigations into the movement of ground were conducted regularly by the Central Research Bureau of Underground Survey, later reorganised into the U.S.S.R. Research Institute of Underground Survey.

The basic method employed in investigating the movement of surface is the pegging out of a network of datum or bench marks on the surface to be studied, and the carrying out of periodic levelling along the datum mark lines (and in some instances also observation of their displacement in the horizontal plane). The data characterising ground-surface movements are compared with the disposition and advance of underground workings, particularly production faces. The results of such observation are taken as a basis for elaborating laws governing the movement of ground depending on the effect exerted by various factors, this being necessary for drawing up well-substantiated projects of safety pillars.

Safety pillars in large mines sometimes contain considerable reserves of valuable mineral. For this reason they should be left only when they are really needed and their size should not be unnecessarily big if unwarranted mineral losses are to be avoided.

Since ground movements over the worked-out areas depend on the properties and nature of rock occurrence, the rules for calculating safety pillars must take account of the conditions prevailing in major mining districts.

We shall now discuss two important examples of erecting safety pillars—in the Donets and Moscow coal fields.

For the Donets coal fields there are four categories of objects (Table 12) that are established by the regulations for protecting surface structures from harmful effects of underground mining.

To design safety pillars, it is necessary to know the spatial position of the surfaces delimiting the ground which has been dislocated. To simplify actual designing, it is usually assumed that these surfaces are planes and, consequently, their spatial position is characterised by an angle of slope to the horizontal plane.

Let us assume that the angle of dip of the seam is $\alpha$ and that a safety pillar has been left in the worked-out area, its upper and

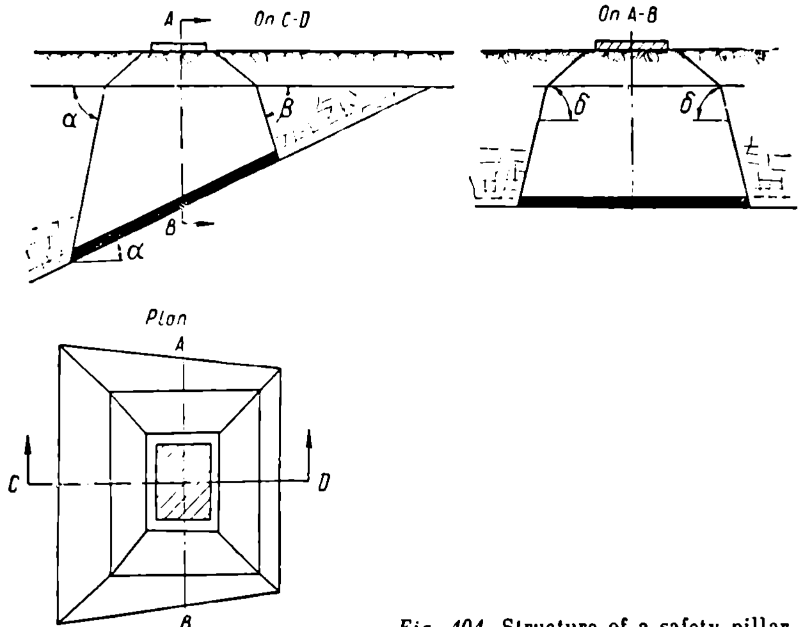



*Fig. 404.* Structure of a safety pillar

lower boundaries extending along the strike of the seam (Fig. 404). The plane delimiting the dislodged ground from the lower side of the pillar, that is, on the side marking the rise of mined-out space, is determined by angle $\gamma$; and from the upper side, that is, on the side marking its dip, by angle $\beta$. If a vertical plane is to be drawn through the centre of the pillar in the direction of the strike, the position of planes delimiting the masses of shifted rocks on the side of the strike is determined by angles $\delta$.

The numerical values of angles $\beta$, $\gamma$ and $\delta$ are determined by the properties of rocks and the angle of dip of the seam. In a mantle rock or drift ground angles $\beta=\gamma=\delta=60°$. When constructing safety pillars in Tertiary and Cretaceous rocks these angles, in view of their horizontal attitude in the geological conditions prevailing in the Donets coal fields, are $\beta=\gamma=\delta=70°$.

For carboniferous rocks the above angles are listed in Table 13.

We have seen that the working seam may occur at a depth where the movement of the ground over the mined-out area does not reach the surface and, consequently, there is no need to leave safety pillars. Besides, it may so happen that the subsidence and sagging of the surface are so slight that they do not represent any danger to surface structures. Accordingly, the depth of a seam (or for that matter any

**Categories of**

| Basic purpose of protection | Categories |
|---|---|
| | I |
| Prevention of mass accidents | Vertical shafts, head frames. Hoist plants. Bridge abutments with spans over 20 m. The River Severny Donets. Water reservoir basins of the Kurakhovka and Volyntsevsky types, dams with their spillway arrangements |
| Prevention of considerable damage and destruction capable of disrupting normal industrial operations | Blast and open-hearth furnaces, foundries, rolling mills, and principal units of glass shops. Coking plants. Coal-dressing mills. Interconnected regional electric power plants and substations. Artificial water reservoirs supplying industrial plants. Coke ovens with recuperation.<br>Power-house boiler plants |
| Prevention of damages causing considerable material losses | Engineering works and buildings of permanent nature and particular importance; 5-storey public buildings and dwelling houses of state importance or distinguished by their architecture |

*Table 12*

**Protected Objects**

of protection

| II | III | IV |
|---|---|---|
| Auxiliary air shafts with no machine hoisting. Inclined shafts. Railway bed of trunk lines. Terminal railway station buildings. Bridge abutments with spans under 20 m. Underground gas pipelines of local importance | Major water conduits, natural and artificial water basins not liable to be drained off; river beds. Ravines with permanent water streams. Air and auxiliary inclined shafts. Local railway lines for general use | |
| Trunk pipelines of regional importance. Pump and water clearing stations. Brick and reinforced-concrete pipes. Electric power stations and substations of local importance. Water cooling towers. Large machine shops. Water-pressure tanks. Pumpworks. Railway locomotive sheds. Railway station buildings and central railway switch posts. Boiler plants and coke ovens without recuperation. Mine fan-houses. Compressor plants. Oil pipelines | Corner masts and cableway stations. Underground rooms with mechanical equipment. Mine locomotive and electric locomotive sheds. Mine machine shops of medium size | |
| 3-4-storey stone public buildings and dwelling houses, permanent medical establishments and schools, irrespective of the number of storeys | Ordinary standard and public 1-2-storey stone buildings in mass construction with bearing walls on continuous footing, with the exception of permanent medical establishments and schools, for example, office buildings of the Chistyakovantratsit Trust, Kuibyshevugol | 1-storey stone buildings, regardless of their purpose, except for schools and hospitals, provided their sides in plan do not exceed 15-20 metres |

*Table 13*

**Angles of Rock Shifts**

| Angle of seam dip $\alpha$ in degrees | Angles of shift, in degrees $\beta$ | $\gamma$ | $\delta$ |
|---|---|---|---|
| 0-5 | 85 | 85 | 85 |
| 6-44 | 90—$\alpha$ | 90 | 85 |
| 45-65 | 90—$\alpha$ | 85 | 85 |
| 66 and above | 100—$\alpha$ but not less than 25 | 85 | 85 |

mineral deposit) over which there is no movement of ground above the worked-out space or, if there is, it presents no hazard for the surface structures, is designated *harmless* or *safe depth.*

Since the intensity of ground movement also depends on whether a given seam is mined by the caving or the filling method the safe depth depends directly on these circumstances too. The regulations governing safety-pillar construction in the Donets coal fields provide for the following methods of determining this safe depth.

*Table 14*

**Factors Determining Safe Depth**

| Angle of dip, in degrees | Classes of protection: I — Rocks with prevalence or presence of thick seams: shales | I — Rocks with prevalence or presence of thick seams: sandstones | II — For any combination of rocks | III — For any combination of rocks | IV — For any combination of rocks |
|---|---|---|---|---|---|
| *For all structures, barring the engineering* | | | | | |
| 0-45 | 350 | 400 | 150 | 100 | — |
| 46 and above | 400 | 500 | 200 | 100 | — |
| *For engineering structures only* | | | | | |
| 0-45 | 300/250 | 350/300 | 150/125 | 100/75 | 50/— |
| 46 and above | 350/300 | 450/400 | 200/150 | 100/75 | 50/— |

First of all, it is determined, depending on the thickness of the working seam, by multiplying its minable thickness $m$, measured normally, by safety factor $k$. Safe depth $H_s$ is measured vertically and equals the product $km$. The recommended values of the safety factor are listed in Table 14 in accordance with the categories of protected objects, angles of dip and the composition of rocks.

The standards listed in Table 14 may be less stringent (that is, they may make it possible to dispense with pillars at smaller depths), provided filling is used.

Safety pillars are designed as follows. In the plan showing the position of protected structures the latter are delimited by contour lines, as far as possible in the form of a rectangle, its sides extending along the strike and dip of the rocks. Depending on the importance of the protected objects, *banquettes* ranging in width from 5 to 15 metres are added to these contour lines. From the final contour lines of the area to be protected planes are constructed at angles $\beta$, $\gamma$ and $\delta$, in conformity with their numerical values. Should the seam occur at a depth, which is less than the safe one, the intersection of these planes with it determines the outlines of the pillar. In the typical example given in Fig. 404 (illustrating a pitching seam) a safety pillar of trapezoidal form will correspond to a rectangular area protected on the surface.

Below is a detailed example of how a safety pillar is calculated and designed, taken from *Regulations for the Protection of Surface Structures*.

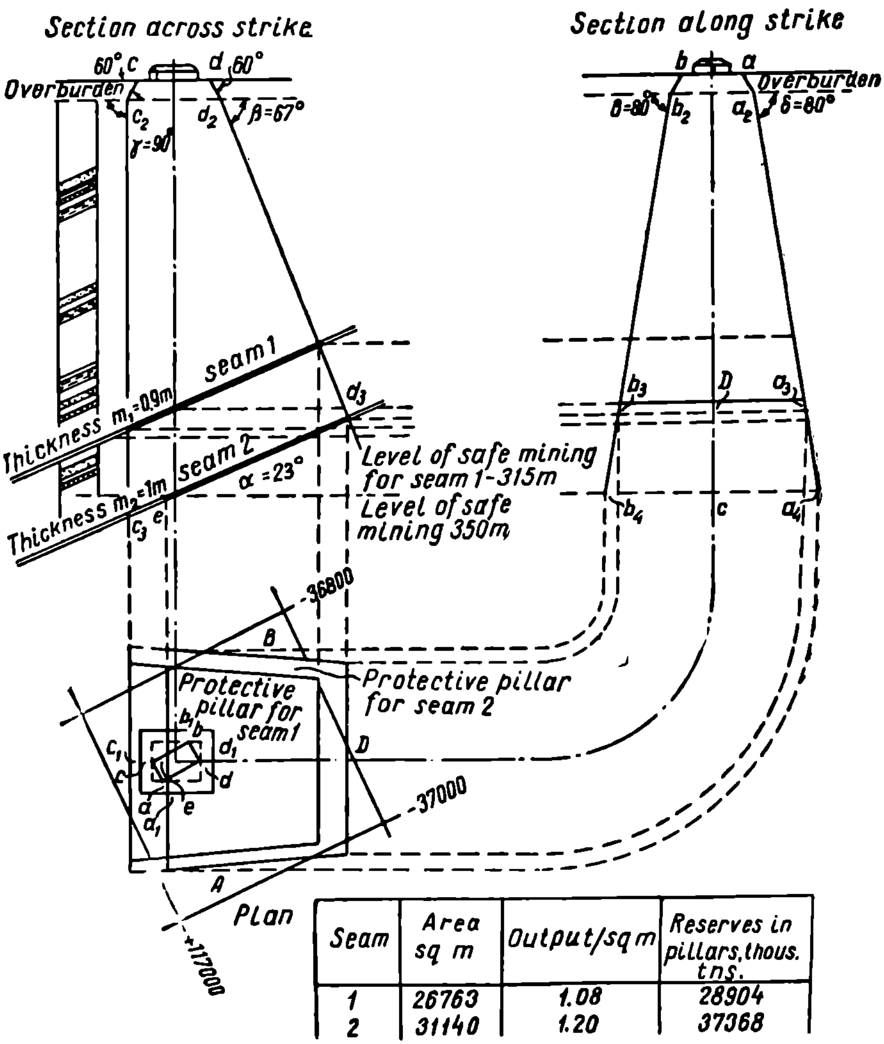
### "*Construction of Pillars for the Protection of Buildings*

"Structure *abcd* (Fig. 405), included in the first category of protected buildings, lies obliquely to the strike of the working seams. The building is 40 metres long and 20 metres wide. The seams dip at angle $\alpha=23°$, the first $m_1=0.9$ metre thick and the second $m_2=1$ metre. The rocks in the strata of the hanging wall are predominantly shales. Carboniferous rocks have a cover of drifts 15 metres thick.

"To plot the pillars in the working seams, it is first necessary to determine the boundaries of the contours of the area to be protected. To do this, we draw lines parallel and normal to the strike through the marginal points of the building—*a*, *b*, *c* and *d*. On the outside of the resultant rectanlge we plot a berm or banquette 10 metres wide. After that we cut a section across the strike through the centre of the contoured area to be protected.

"On this section, from the contour lines of the protected area, two lines are drawn in the drifts at an angle of 60° to the horizontal. Further, line $c_2c_3$ is drawn through the point of intersection of the first two lines and carboniferous rocks (points $c_2$ and $d_2$ on the section) at an angle of $\gamma$ which in this instance equals 90°, and another, $d_2d_3$, at an angle of $\beta$, equalling $90°-\alpha=67°$. The intersection of line $d_2d_3$ and the coal seams determines the upper boundary of the pillars on the section. Their lower boundary can be determined either by the intersection of line $c_2c_3$ and the seam, or by the intersection of the seam and the "safe" depth elevation.

"The safe depth of mining is $H_1=0.9\times350=315$ metres for the first seam and $H_2=1.0\times350=350$ metres for the second.




| Seam | Area sq m | Output/sq m | Reserves in pillars, thous. tns. |
|---|---|---|---|
| 1 | 26763 | 1.08 | 28904 |
| 2 | 31140 | 1.20 | 37368 |

*Fig. 405.* An example illustrating the structure of a safety pillar

"Since the point of intersection of line $c_2c_3$ and the first seam lies above the safe depth elevation for this seam, the lower boundary of the pillar in it will run through this point of intersection.

"In the second seam the lower boundary of the pillar will go through the point of its intersection with the safe depth elevation ($H_2$=350 metres) because line $c_2c_3$ intersects the seam below this elevation mark.

"Further on, a section on strike is cut on the same scale as that made across. From the outlines of the protected zone lines $bb_2$ and $aa_2$ are drawn in drifts at an angle of 60°. From resultant points $b_2$ and $a_2$ inclined lines are run in carboniferous rocks at an angle of $\delta=80°$ until their intersection with the seams at elevations corresponding to the upper and lower boundaries of the pillars. The resultant intersection points determine the size of the pillars on strike. The plotted contour lines are shown in Fig. 405."

If the protected object is built on a plot extending in one direction—for instance, a regular railway line which, moreover, runs obliquely to the strike of the deposit—the construction of a safety pillar will prove to be much more complex and must be plotted by separate sections.

In the mining of coal measures, determination of a safe depth should take into account the aggregate thickness of all the seams in a measure and the nature of their occurrence.

In the Moscow coal fields the attitude of seams is much simpler than in the Donets basin. They are shallow-seated, almost flat and, with rare exceptions, are extracted one by one. The angles of rock movement on all sides of a safety pillar have to conform to two classes of protected objects—the more important (category I) with an angle of 45° and category II with an angle of 55°.

As stated earlier, abandonment of safety pillars entails losses of mineral in the earth's crust. Therefore, it should be done only when it is really necessary for the protection of surface plants, water reservoirs, etc. In designing mines and building surface plants one must always strive to arrange these objects in a manner obviating the necessity of protecting them by safety pillars, provided this is feasible technically and economically. This is of particular importance in working large deposits, especially if they contain highly valuable minerals. In mining thick, steeply dipping mineral deposits, for instance, the shafts and surface plants should be located in the foot wall; water reservoirs should be drained off or deviated from the area of the expected draw in order to obviate the need for safety pillars.

In all circumstances the size of safety pillars should be restricted to a bare minimum. In strong minerals and enclosing country rocks safety pillars may be latticed, that is, be partly extracted by narrow face headings. In other words, in such instances one may make use of the principle of natural support pillar mining. In ordinary solid safety pillars well-supported mine openings can be driven.

When the mine is abandoned, safety pillars can be recovered, insofar as this is technically possible in local conditions.

CHAPTER XXIV

# CLASSIFICATION AND CHOICE OF MINING METHODS

## 1. Programme of Study and Description of Mining Methods

To give a sufficiently full characterisation of a mining method actually employed or planned, one should study it and, if necessary, describe it in many of its aspects. The pertaining data and characteristics may be classified into the following groups.

### *I. Mining and Geological Conditions*

Name of the mineral extracted. Shape of the deposit—bed, sheetlike deposit, placer, vein, etc. Thickness of the deposit (true and lateral), prevalent thickness, maximal and minimal deviations. Angle of dip and its variations. Pitch or hade of the ore body. Area of the horizontal section of the deposit. Typical geological faults and disturbances in the attitude. Matter composition of the minerals. Distribution of useful components in the deposit. The nature of the contact between the deposit and the country rocks surrounding it. Hardness, jointing and firmness of the mineral. Its density (volume or unit weight). Size distribution and ability to compact. Petrographic characteristics of enclosing country rocks. Their hardness, jointing and stability. Abundance of water in the deposit. Properties of the mineral's self-ignition. Oxidability of the ores subject to flotation. Evolution of noxious gases. Harmful properties of dust (explosiveness of coal and sulphide dust, dangerous properties of quartz dust with respect to silicosis).

### *II. Mining Characteristics of the Method To Be Adopted in Working a Deposit*

Designation of the mining method. Level interval. Sublevel interval. Extent of the working section or block on strike. Size of pillars and support pillars. Slice thickness. Cross-section and support of development openings. Sequence of driving development openings. Advance rate of faces. Rate of development headings advance over that of production faces. Order of recovery of mineral reserves in a working section or block. Order of mining sublevels, pillars or slices. Shape and size of working faces. Their interlocation. Advance direction of working faces. Method of stoping. Methods of controlling enclosing rocks: by support pillars, timbering, filling (complete, partial), shrinkage-stoping, caving (spontaneous or induced), or by different combinations of these methods. Methods adopted for the delivery of the mineral to the haulageway. Scram and grizzly levels (in ore mining). Ventilation of development and productive workings. Lighting of mine workings. Measures envisaged by the system of mining against penetration of water and inrushes of water-bearing rocks. Preventive measures against underground fires, as part of the mining method adopted.

### III. Mechanisation of Mining Operations

Brief specifications of machines (trade-mark, capacity, ratings of driving motors and overall dimensions) used for drilling holes, undercutting and breaking the mineral or country rocks: drilling machines, electric augers, drill-wagons for the underground boring of deep blast-holes and large-diameter holes; coal cutters, cutting and loading machines (combines), coal planers, hydraulic giants (in hydraulicking), etc. Analogous information on machines and equipment employed for the transportation of the mineral and waste: conveyers, scrapers, loading machines, mine car spotting tugger hoists, district electric locomotives, capacity and overall dimensions of mine cars. Machines and mechanical plants for ventilation and mine drainage (needed for the system of mining).

### IV. Organisation of Work

Organisation of operations in the faces of development and productive workings. Graphs (planograms) of cyclic operation and labour distribution charts (number of miners, number engaged in each shift and classes of work performed). Advance rate of faces per cycle or round. Number of cycles or rounds per day and per month. Coordination of all operations for the whole of the producing section (linked with the system of mining).

### V. Technical and Economic Characteristics of the Mining Method

Mineral output in productive and development workings per day per shift, in individual faces (walls, stopes) and in the section as a whole. Monthly tonnage produced by the entire section. (If, because of features specific to the adopted method, the tonnages tend periodically to vary widely, for instance, during shrinkage-stoping and subsequent drawing of the ore, the characteristics above must be given for individual stages of mining.) Yield of the mineral from development and working faces (for the whole of the mining method in per cent). Mining or working losses of the mineral (in per cent). Degree of dilution (in per cent to the total content of valuable components in undiluted ore). Time required for the recovery of the aggregate reserves in the working section or block. Output per faceman and per miner for the whole of the section (in tons of mineral, or in cu m of ore, or ore and barren rock together per shift). Explosives consumption per ton, or per cu m in grammes. Mine timber consumption per 1,000 tons or 1,000 cu m of the mineral, or the aggregate amount of the mineral and waste. Electric power and compressed air consumption per ton or cu m. Mining cost of a ton or cu m of the mineral, or of the mineral and waste together for the whole of the producing section, including delivery to the haulageway.

## 2. Classification of Mining Methods

Methods employed for mining solid minerals in deposits may be divided into the following principal groups:

I. Underground mining.
II. Surface mining.
III. Combined mining.
IV. Special methods of mining.

No explanations are needed for singling out methods I and II. One example of method III is mining by glory holes (milling) (see

Chapter XXI, Section 14), when the mineral is extracted by the opencast method and loaded into transport vehicles and subsequently hauled in underground workings.

Special methods include those in which actual mining is characterised by changes in the *native* (*aggregate*) *state* of the extracted mineral. They include underground coal gasification, ore-mining by underground leaching, extraction of sulphur through boreholes by evaporation, etc.

As we have seen, the systems used in mining solid minerals by the underground method vary widely and are frequently complex. For a more or less full *characterisation*, one should refer to many of the features enumerated above (Section 1).

However, the *classification* of mining methods cannot be founded on all the above-cited, extremely numerous features. Their classification should be based only on the especially important and typical features, according to which it is advisable to divide and single out the systems of mining.

Most of the hitherto proposed classifications of mining methods were based on methods of controlling enclosing rocks and on the arrangement of development openings.

It is noteworthy in this connection that the division of mining methods into groups according to the arrangement of development openings is generally adopted both for drawing up classifications and for working coal and other sheet deposits, whereas the classifications for the systems applied in mining ore deposits are founded on the second principle—that involving the method of enclosing-rock control. This difference in the approach to the characteristic features, on which the classification is based, is by no means accidental or one chosen arbitrarily by the compilers of the classifications, but is explained by the fact that for the sheetlike deposits the arrangement of development openings is very typical and at the same time simple and convenient because of their regular shape. That cannot be said, however, of ore deposits whose shapes are on the whole irregular, both generally and in particular cases. Because of this, the location of development openings in each concrete case is less systematic and, at any rate, more complex than in coal deposits. At the same time the problems of rock-pressure control here can be solved much more easily and in a greater variety of ways. It is these reasons that prompt the classification of the mining methods employed in working coal and other sheet deposits in accordance with the spatial arrangement of development openings, and those used in extracting ore bodies by the method of enclosing-rock control.

The author favours the following classification of mining methods in working solid useful minerals:

## CLASSIFICATION OF METHODS AND SYSTEMS IN MINING SOLID USEFUJ MINERALS

*Note*: Asterisk denotes systems little used.

### I. Underground Mining

#### *A. Sheet Deposits*

a) Methods of mining without division into slices (nonslicing systems of mining):

1. Continuous (longwalls): on strike; to the rise.
2. Pillar mining: long-pillar method; * pillar-and-stall method; long-pillar method up raise; shield-mining method.
3. *Room mining.
4. Combined methods: room-and-pillar system; twin-entry method.

b) Slicing methods of mining: horizontal slicing; inclined slicing; * transversely inclined slicing; * diagonal slicing.

#### *B. Nonbedded Deposits*

Systems of mining with enclosing-rock control:

a) Methods involving abandonment of natural support pillars: continuous breast stoping; pillar mining; room-and-pillar method: sublevel method.

b) Artificial support: stull-set method of mining; square-set method of stoping.

c) Filling method.

d) Shrinkage-stoping.

e) Caving of capping: horizontal top slicing; * inclined top slicing.

f) Caving of ore: sublevel caving; induced block or bulk caving; spontaneous (uncontrolled) block or bulk caving.

g) Systems of mining with combined methods of enclosing-rock control.

### II. Surface Mining

### III. Combined Underground and Surface Method of Mining

* Glory-hole mining (milling).

### IV. Special Methods of Mining

* Underground gasification of coal; * underground leaching of ores; * underground dissolution of salts; * underground evaporation of sulphur.

The classification of mining methods above has been compiled on the basis of the following considerations.

To make it simple and easy to understand, each classification must be based on a limited number of the features most specific to it. In the above, such features are spatial arrangement of development openings for sheet or bedded deposits and methods of enclosing-rock control for nonbedded mineral bodies.

The characteristics of the stoping method should not be regarded as a classification feature, for the actual stoping can be effected in a great variety of ways with one and the same system of mining. For instance, there is no doubt that long-pillar mining is an independent system. At the same time, however, actual extraction of the mineral in the pillar may be done in a number of different ways—by continuous faces, slab or open-end entries, by strips, blasting operations, pneumatic hammer drills, coal cutters, cutting-and-loading machines, coal planers, etc., and in working placers also by means of "surface thawing", etc. Here is yet another example. Singling out sublevel caving as an independent system of mining is not challenged by anyone, and this quite irrespective of whether stoping is effected through slab drifts,"open rooms", or any other method. The methods of stoping are of importance for the characterisation of a mining system and its operative results, but this does not mean that they should be regarded as a feature indispensable for the classification of any mining method. And this is all the more justified because within the framework of one and the same system it is possible to switch over from one method of stoping to another, a thing that is now practised quite frequently, particularly thanks to the rapid progress of mechanisation.

## 3. Choice of Mining Method

The choice of a method for mining a deposit is influenced by numerous factors, discussed in Chapters IX and XIX.

The description of the principal methods of mining enumerated the conditions most suitable for each. But since the combinations of diverse factors influencing the selection of a mining method may be extremely variable, this choice for a particular deposit is complicated by its geological features, as well as by the mining and economic sutuation.

The method selected must meet the basic demands of the conditions in which it is called to operate (see Chapter VII).

In a very general outline the method of the choice itself boils down to comparing the features of each one of the mining systems which may possibly be employed in actual geological, mining and economic conditions. To make such comparison sufficiently systematic and preclude any possible faulty judgement, K. Charkviani has suggested a *method of elimination*. Essentially, this method implies consecutive elimination, after a pertinent analysis, of all systems of mining whose application in given conditions falls short of the necessary requirements.

To facilitate this procedure of elimination, Charkviani has drawn up special tables for working nonferrous metal ore deposits. The elimination procedure usually reduces the number of mining systems to one, sometimes two and rarely three, which can be employed in given conditions. If there is only one system, the choice is final; if there are two or three methods capable of competing with each other, the ultimate decision is taken after a thorough technical and economic comparison.

*Part Three*

# OPEN-CUT MINING

CHAPTER XXV

# BASIC DEFINITIONS AND TERMINOLOGY

## 1. Conditions Warranting Open-Cut Work

Open-cast mining of minerals is justifiable technologically and economically when they lie immediately near the surface, or at a relatively small depth. In this connection it should be added that, because of the considerable progress made by mechanisation, minerals can be quarried out to advantage at ever-increasing depths (see Section 2).

Fig. 406 outlines an open-pit layout, where a large body of mineral *a* is directly exposed over a more or less extensive horizontal surface, sometimes slightly covered by overburden *o*. Instances of such nature are not infrequent in the open-cut mining of such building materials as stone, sand, clay, etc.

Fig. 407 depicts a pattern adopted for mining large masses of mineral *a* on a mountain slope.

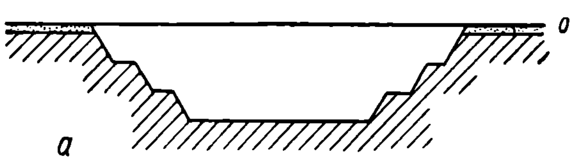



*Fig. 406.* Open-cut work layout in the absence of overburden

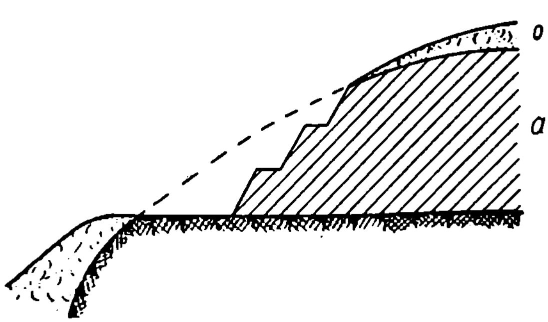



*Fig. 407.* Open-pit layout on a mountain slope

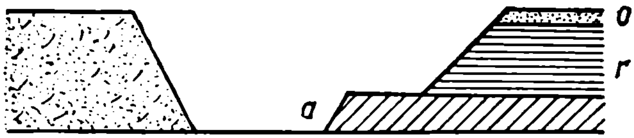



*Fig. 408.* Open-pit layout on a flat bed with horizontal ground surface

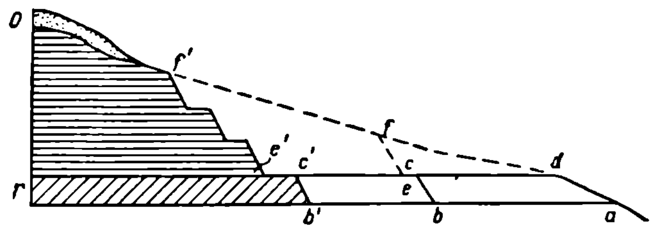



*Fig. 409.* Open-pit layout on a flat bed with sloping ground surface

Open-cut work is also possible when the deposit is not exposed directly at the surface, but is covered by considerable amounts of overburden or country rocks, whose thickness should not exceed a certain limit. In cases like this the deposit may occur either horizontally (Fig. 408) or at any angle to the horizontal (Fig. 409).

The production unit of a mining enterprise extracting the mineral by the open method is called *open pit.* This term is analogous to mine, which extracts the mineral underground. It should be noted that the term open pit often also designates an *open-mine working* in which the mineral is excavated by the open-cut method.

An open pit (quarry) in which coal is mined is often called *open cut.*

## 2. Determining the Depth of Open-Cut Work

To extract the mineral by an open-cut method it is necessary preliminarily to remove certain amounts of barren rocks. This operation is called *stripping* and the ground removed *overburden* or *spoil.* Of importance in this connection is not only the absolute amount of the ground subject to removal, but its relative volume per unit of the mineral extracted. It may, for instance, prove impractical to strip cover-rock strata 15 metres thick from a coal seam, if the seam itself is 1 metre thick, but it might be economical if the thickness of tho seam is as much as 5 metres. The ratio of the

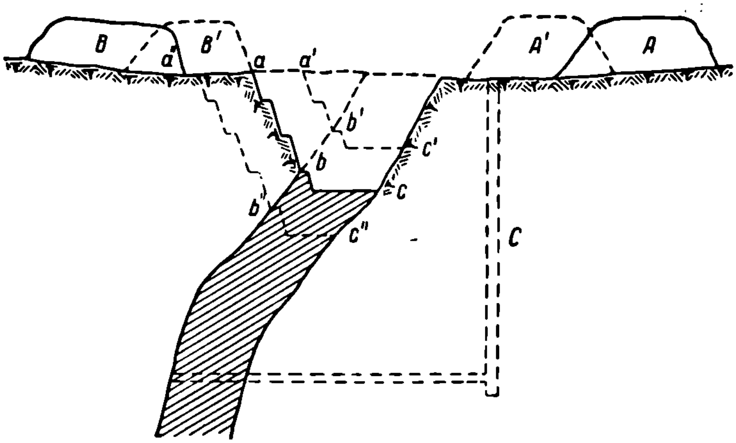



*Fig. 410.* Pit layout in steep dip

overburden volume to the amount of mineral reserves already stripped or to be stripped, expressed in volumetric (or weight) units, is called *stripping ratio.*

When the earth's surface and the mineral occurrence are more or less flat, the stripping ratio is fairly uniform (see Fig. 408). When the surface slopes, this ratio alters along with the increase in the size of the open pit, even if the deposit lies horizontally and is uniformly thick throughout (Fig. 409). In working an inclined or steep seam of a more or less uniform thickness, the relative amount of overburden to be stripped increases with the depth of the open pit (Fig. 410). Thus, to excavate volume *c'cbb'* of the mineral it is necessary to remove volume *aa'b'b* of the hanging wall rocks during the stripping process, while in deepening an open pit to excavate a similar volume of mineral *bcc''b''* an appreciably greater volume of barren rock *aa''b''b* has to be removed by stripping.

Similarly, in the case of a horizontal or flat occurrence and increasing thickness of the overburden (Fig. 409), to excavate a given amount of mineral *abcd*, the volume of the overburden to be stripped will be *efd*, but subsequently the extraction of the same volume of mineral *b'c'cb* will necessitate stripping a larger volume of capping—*c'f'fe*.

For conditions prevailing in each open pit there exists a *maximum proportion* of the amount of waste or spoil removed to a unit measurement of the mineral, which it would be uneconomical to exceed in the present stage of technological development. When this limit is reached and the deposit lies much deeper still, it is more profitable to change over to underground mining, through a vertical shaft, for example ($C$ in Fig. 410). This maximum proportion depends in general on cost $a$ charged against mining by the open-cut method of 1 cu m of mineral, cost $b$ of overburden stripping per 1 cu m and the *stripping ratio*. By using these symbols, we find that the total cost of mining 1 cu m of mineral by the open-cut method, including the cost of stripping operations, will amount to

$$a + bx. \tag{1}$$

Conditions economically justifying the use of the open-cut method instead of underground mining will then be determined by the formula

$$a + bx < c \tag{2}$$

where $c$ is the mining cost of 1 cu m of mineral by the underground method.

The *maximum stripping ratio* can be found from an equation expressing the cost of mining by the open-cut and underground methods

$$a + bx = c,$$

whence

$$x = \frac{c-a}{b}. \tag{3}$$

Depending on the geological structure of the deposit, the size of the open pit, the available machinery and organisation of work costs $a$, $b$ and $c$ may vary widely and, consequently, the differences in the value of the maximum stripping ratio are also liable to fluctuate appreciably.

If, depending on the geological structure of the deposit, the stripping ratio tends to mount along with the increase in the depth of the pit, the maximum stripping ratio must correspond to an economically profitable *maximum depth of the pit*. As an example take Fig. 410, which clearly illustrates the progressive growth of the relative amount of overburden that has to be removed as the pit is worked in depth.

Emphasis should be laid upon the fact that the maximum stripping ratio and the depth of the pit corresponding to it have to be established from the stipulated equality between the mining cost by open-cut and underground methods not for the whole of the pit, *but for the particular level or elevation at which this equality becomes valid*. This very important stipulation is explained as follows. Let us assume that volume $c''b''bc$ (Fig. 410) of the mineral is the last in depth whose open excavation justifies removing volume *b''a''ab* of the overburden, for in these conditions the cost of mining by the open-cut and underground methods is the same. But to make possible open-cut excavation of the same volume, *bb'c'c*, of the mineral lying immediately over the first, a smaller volume, *baa'b'*, of cover rocks has to be stripped and, consequently, the aggregate cost of mining 1 cu m of the mineral in this case is lower. This applies all the more to working of mineral occurring still nearer the surface. Hence, if we assume that elevation *c''* is the maximum economic depth of the pit, mining cost per cubic metre (or ton) of the mineral in the pit will *generally* be lower than that by the underground method. In other words, if the maximum depth of the pit were established on the basis of the stipulated equality between the average unit mining cost of the mineral for the whole of the pit, on the one hand, and that by underground method, on the other, the pit's maximum depth would be greater than in the first case and the mineral reserves lying below elevation *c''* would cost more to excavate in the pit than by the underground method. The economic disadvantage here is quite evident. From the economic standpoint, therefore, the maximum depth of the pit should be established at a level where the open-cut and underground cost of mining is the same.

Hence the maximum profitable depth of the pit, that is, *the boundary line between open work and underground mining*, may be estimated as follows.

Taking as a basis the height of the banks and their arrangement, we draw a series of cross-sections for the planned pit at different elevations. On each profile these elevations can be conveniently brought to the bench levels (as shown in Fig. 410). From these cross-sections one can estimate, for each elevation and according to the number of benches: 1) the volume of the mineral; 2) the amount of the overburden to be stripped—if the rocks constituting the overburden are sharply distinct in nature (for instance, drifts and bedrock), the estimates have to be made separately for each type of cover rock; 3) stripping ratio; 4) mining cost per 1 cu m of overburden and mineral, and, lastly, 5) total unit mining cost by open-cut mining, including the cost of stripping. The results of these computations are tabulated (Table 15).

The lowest level of the pit is the one where the cost by open work is the same as that by underground mining.

For the sake of clearness the data listed in Table 15 may be depicted graphically (Fig. 411).

In order to facilitate and speed up these rather labour-consuming calculations (particularly if the shape of the deposit is complex), V. Rzhevsky has suggested a spatial graphic method. In some cases

*Table 15*

**Estimation of the Maximum Depth of the Pit**

| Pit sections between levels | Volumes, cu m | | | Stripping ratios | | Mining cost per cubic metre | | Cost per 1 cu m charged against 1 cu m of the mineral mined | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | Overburden $V_h$ | Bedrock $V_b$ | Mineral $V_m$ | $V_h/V_m$ | $V_b/V_m$ | Overburden $q_h$ | Bedrock $q_b$ | Overburden $Q_h$ | Bedrock $Q_b$ | Mineral $Q_m$ | Total $Q$ |
| *0-1* | | | | | | | | | | | |
| *1-2* | | | | | | | | | | | |
| *2-3* | | | | | | | | | | | |
| .... | | | | | | | | | | | |
| .... | | | | | | | | | | | |
| *n-1-n* | | | | | | | | | | | |

the maximum, economically justifiable depth of the pit can be determined more quickly by the analytical procedure. The essential point of the method is that the aggregate cost of open-work mining of a cubic metre (or ton) of the mineral is found as a mathematical function of the pit depth. The equation arrived at by comparing this cost with the cost of underground mining gives us the depth

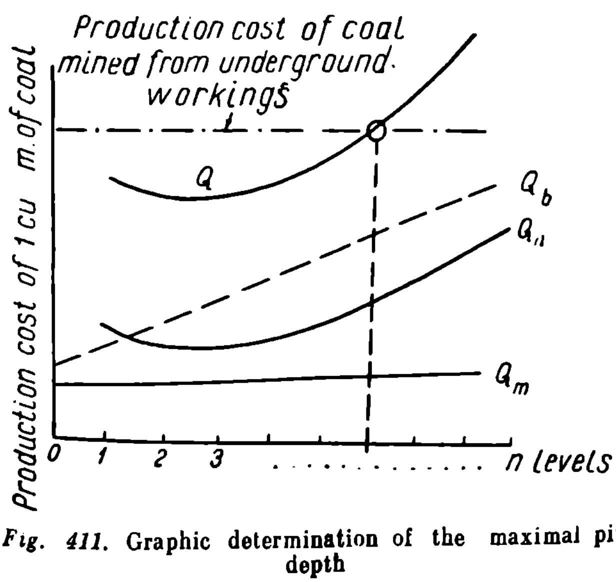



*Fig. 411.* Graphic determination of the maximal pit depth

we are looking for. But such an equation can be arrived at only if we assume that the thickness and the attitude of the deposit follow a definitely regular pattern, which is not always the case. Because of that the use of the analytical method of determining the pit's depth is rather limited.

Both in tabular and analytical methods of determining the pit's depth particular attention should be attached to preliminary estimates of the cost of a cubic metre of overburden stripping and actual mining of the mineral. Depending on the local natural, as well as technical and operating conditions, these costs may vary extremely widely. In view of this, one should not, generally speaking, rely too much on the statistics on the existing open-cut works, but determine these costs in each individual case by detailed calculation.

Because of the sharply changing properties and attitude of the deposit, the depth of a pit may not be the same in all its sections. In such instances the depth of each section is calculated separately. The ultimately acceptable depth of the pit is chosen with due consideration of the conveniences and economic advantages it offers for the transportation system throughout the entire open-cut works.

If the deposit is horizontal or dips slightly, but the capping is of irregular thickness (see Fig. 409), it becomes necessary again to establish the bounds of the open-cut work. The deposit is then divided on the plans into several sections (*ab*, *bb'*, etc.), and the estimates are made as above.

For this it is necessary to have good topographic plans with contour lines plotted on them, as well as plans with isometric lines showing the attitude of the bottom and the back of the deposit, its thickness and the roughly estimated spatial distribution of the useful mineral.

Since the maximum stripping ratio may be quite different, depending on local natural and technical conditions, the practice of taking "current" figures for stripping ratios as a guide is to be ruled out altogether. If one follows the pattern of calculation referred to above, the maximum stripping ratio should be determined in each concrete case. In general, the progress of mechanisation in open-pit mining *makes it possible continuously to increase this ratio.*

In conditions favourable for the development of open-cut mining the pits may be very deep. In the Krivoi Rog district, before it went over to underground mining, the pits were as much as 130 metres deep. One brown coal pit at Korkino (Urals) is at present 210 metres deep, and it is planned to deepen it still more. The projected depth of a copper ore pit at Kounrad (Kazakhstan) is 240 metres.

In many instances the depth of a pit is determined by a shallow occurring bottom of the deposit (for instance, open cuts in the Cheremkhovo coal fields, the Sokolov bauxite pit in the Urals and others).

There have been cases of the danger of landslides on the slopes of a pit prompting to go over to underground mining earlier than stipulated by the project (for example, one of the open-coal cuts at Yemanzhelinka in the Urals). And conversely, special circumstances sometimes make it imperative to continue open-cut mining at depths where the underground extraction of the mineral would seem more profitable economically. A typical example is the mining of thick beds of self-igniting coal.

There are some noteworthy examples of open-cut mining abroad.

The Bingham pit in Utah, U.S.A., is 480 metres deep, 1,200 metres wide and 1,800 metres long. Its annual output is over 35 million tons of copper ore, the average per man per shift being 74 tons of mineral and overburden. The highest annual output of iron ore in an open pit in the United States is 6,000,000 tons.

The Chuquicamata open pit in Chile, which is about 300 metres deep, produces 33 million tons of mineral and rock, including 18 million tons of copper ore, with output per man per shift amounting to 58 tons.

### 3. Advantages and Drawbacks of Open-Cut Work as Compared to Underground Mining

Contrasted with underground mining open work presents many *advantages*.

1. The principal advantage is that the efficiency of labour is considerably higher in open pits, while the cost of mining is lower. In coal industry a miner achieves 3-4 times as much in open-cut mining as he does underground. That is because it is possible to use highly efficient mining equipment and transport facilities, there being no limits to the overall dimensions of machines, as is the case in underground workings.

2. Large working faces and the use of highly efficient machines for the open mining of large deposits yield considerable tonnages.

The same factors make it possible to increase output in a newly established open pit much faster than in an underground mine. The reconstruction of an active open pit enables to raise production more quickly than is the case with an underground mine.

It is noteworthy in this connection that these advantages were exploited to the maximum and with exceptional success in mining coal deposits in the Urals during the Great Patriotic War.

3. Open pits may have a shorter service-life than underground mines, since here it is possible to transfer the costly main items of mining and transport equipment and machines to other pits for further use. Hence, pits working even limited reserves may yield large annual tonnages. The open coal cuts to be put into operation in the

next few years will have an average annual capacity of 1.6 million tons.

4. Open pits need no support, filling, ventilation and artificial lighting (except at night).

5. The percentage recovery of the mineral is higher in open work. True, a certain amount of it may be lost together with the barren rock of the partings left in the pit, in sink holes in the bottom of the deposit and during haulage (spillage). All this makes it imperative to take systematic steps to prevent unwarranted losses in open-pit mining too.

6. When necessary, large-size monoliths for construction and sculptural purposes can also be quarried.

7. Working conditions in the open, in good weather and not too severe climate, are better than underground.

8. There is less danger of accidents. However, the presence in the pits of numerous machines, the wide-flung transport network, blasting operations, the possibility of mineral and rock blocks falling from the faces are a constant source of potential accidents. Strict observance of safety rules and their enforcement, therefore, are just as imperative in the open-cut pits as underground.

The *disadvantages* of open-cut mining are:

1. Rainfall and severe cold make work in open pits rather difficult. Large-scale operations are conducted the year round, but in severe winter conditions labour and machine efficiency drops considerably.

2. Well-developed open-cut work requires considerable tracts of land for pits and barren rocks, with the result that it is often lost to agriculture.

3. Open work at night requires large areas to be provided with artificial lighting.

4. When large coal beds are excavated by the open-cut method, the pits at first yield lower-grade coal in the weathering zone. This decayed coal may, for one thing, be devoid of coking properties, even though normally coal in a given bed is capable of coking.

These drawbacks of open-cut mining, however, are outweighed by its advantages over the underground mining method. Therefore, in conditions enumerated in Section 1, preference should be given to the open-cut method.

It was originally held that the organisational pattern of open-pit work was simpler than that of underground mining. That, however, was true only of small and primitively equipped pits. Large modern open pits, with their heavy-duty electric and transport machinery and installations, require the elaboration of detailed mining plans, more comlex than those needed for underground mining.

We have seen above (Chapter XI) that the organisation of underground mining based on the cyclic principle operations is of a paramount importance for the proper conduct of work. Similarly, *technological* or *operative charts* (*graphs*) can be compiled to provide for detailed planning of continuous operations in open-cut mining too.

In large open pits *masses of barren rock* and minerals have to be excavated and moved. At the Korkino lignite open pits or the Magnitogorsk iron ore pit, for example, many thousand cubic metres of overburden and mineral are mined and moved every day. That is why *complex mechanisation* of all the operations involved is just as essential in open-cut mining as it is in underground working.

At the same time, as pointed out above, open-pit mining does not restrict the choice of the size and type of machines, a factor that has to be reckoned with in underground mines (small cross-section of mine openings and stoping area, availability of timbering, evolution of methane, presence of explosive dust). Therefore, the machines and equipment used in open-cut mining may be larger in size.

The successes scored by the Soviet machine-building industry constantly expand the sphere of the efficient application of open-cut mining and, consequently, contribute to its growing importance in the national economy of the country.

There are numerous open pits producing coal, iron and copper ores, limestones, phosphorites and other minerals, as well as those engaged in working placers, notably gold placers, and some of these pits have high production capacity. It has already been mentioned that a number of deposits, formerly worked by underground methods, can now be extracted much more economically by the open-cut mining method.

Construction work carried out on an immense scale in the U.S.S.R. requires considerable amounts of building materials and these are almost exclusively obtained from quarries.

## 4. Bench Mining

Both the stripping of the overburden and the extraction of the mineral in open pits is generally effected in benches or banks (Fig. 412) to ensure easier and safer work at the faces. A bank includes the following elements: slope *a*, bottom or lower berm *b*, top or upper berm *c*, the height of bank *h* and slope angle β. Line *d*, marking the intersection of the slope and the berm of the bank is called *edge of the bank*.

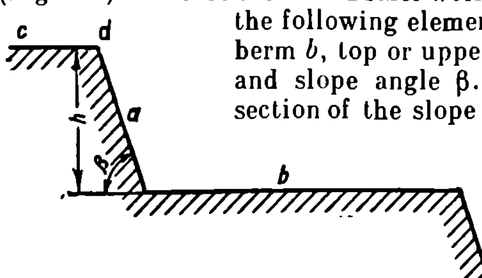



*Fig. 412.* Bank in open-cut mining

The *height* of a bank varies greatly. Up to a certain point a considerable height of banks has major advantages: 1) it facilitates the work of transport, since there are fewer banks and berms (benches) with railway tracks in this case; 2) it minimises the amount of inefficient ripping and levelling out of the bank bed, which is of special importance in working hard rocks. If the bank's height is below that conforming to working dimensions of the power shovel, the latter's efficiency is impaired.

On the other hand, a progressive increase of the bank's height is fraught with the following disadvantages: 1) greater probability of blocks and lumps of the mineral and rocks rolling down the slopes of the banks because of jointings, slabbing and slides; 2) in falling from a bank, a piece of rock, even a small one, gathers momentum and may cause a serious injury; 3) examination of a bank slope for overhanging slabs becomes difficult, particularly at night; 4) blasting strong rocks with large explosive charges put into deep holes may, if the banks are high, yield excessively large blocks of rock (oversize), which cannot be loaded into transport vehicles and have to undergo expensive and difficult secondary breaking; 5) if the geological structure of the deposit requires selective (separate) mining, the complexity of the operation increases with the height of the bank.

These relative advantages and drawbacks of high banks manifest themselves in different ways, depending on the hardness of rock. In determining a bank's height, one should bear in mind the type and size of power shovels to be employed. With shovel-excavating ground requiring no preliminary blasting, the banks are from 4 to 10 metres high and rarely any higher. The weaker and the more friable the rock, the smaller the bank's height. When stronger rock is mined, one requiring preliminary blasting, the shovel is used almost exclusively for loading blasted ground into railway cars or other transport vehicles. In such instances the banks are 10-15 metres high, and occasionally even slightly higher.

With chain-and-bucket excavators the height of the banks depends on the size of machines and the method of work adopted.

The height of the banks cut in the useful mineral occurring at a gentle dip often depends on the thickness of the deposit. For example, in the open-cut mining of coal in the Urals rock banks are usually 10-15 metres high, whereas the height of the banks cut in coal ranges, depending on the thickness and attitude of the seam, between 5 and 25 metres. The height of the banks at the open-cut iron ore pits in the Urals is 10-12 metres at the Magnitogorsk mine, 16 metres at the Gora Blagodat mine, 16-28 metres at the Bakal mine. Limestone at the Big Yelenovka quarry (Donets basin) is mined in banks 12 metres high.

The *slope* of the bank must be somewhat inclined in relation to the berm. In other words, the *angle of slope* should be less than 90°. The weaker the rock the flatter the slope. In the case of friable and soft rocks, the angle of slope should not exceed that of the angle of repose of the rock to be excavated.

To a large degree, the stability of the slope depends, in addition to the petrographic composition of the rock, on the jointing and bedding planes of the rock and the extent to which it is saturated with wa-

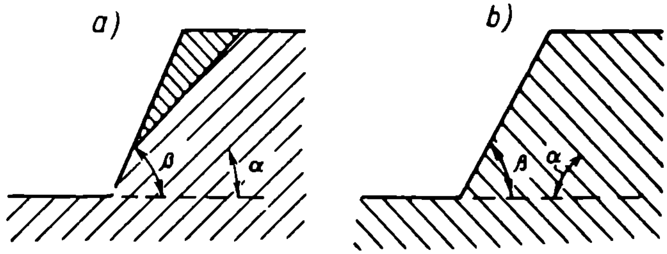



*Fig. 413.* Effect of bedding on bank-slope stability

ter. The effect exercised by bedding planes or jointing is illustrated in Fig. 413. If (Fig. 413*a*) the angle of slope β and that of dip α (or the angle of jointing) are oriented in one and the same direction, prisms *a* may slide down along the bedding planes or jointings. All other conditions being equal, the slope will be steadier if its inclination and the dip of rocks are oriented in opposite directions (Fig. 413*b*).

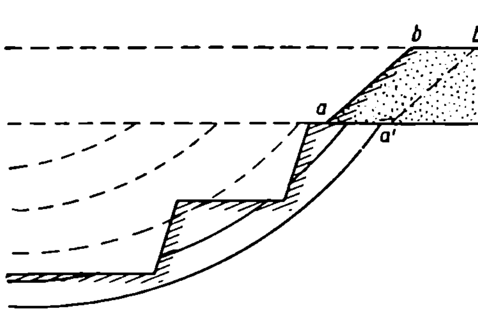



*Fig. 414.* Hazard of slides in an open pit working a syncline limb

The probability of slides along the bedding planes increases when water penetrates through them (or through fissures). Characteristic of open-cut mining of coal deposits is the case illustrated in Fig. 414, in which it is a limb of a syncline that is worked out by a pit. In these conditions, slides along the bedding planes are quite possible. To preclude the danger of such slides, the slopes of the banks should be shifted from position *ab* to position *a'b'*, the angle of their inclination flattened, surface water diverted, and adequate drainage of underground water provided for (Chapter XXVI, Section 12).

In dry mantle rocks (drifts) containing sands and clay, the angles of slope are set at 40-50°. If this overburden ground is aquiferous, the angle of slope becomes flatter as the water abundance increases. The angle of slope in a ledge ground usually varies from 50 to 70°, depending on its hardness, and is occasionally somewhat greater. In aquiferous bedrocks this angle is flattened down to 5-10°.

The magnitudes of the angle of slope given above are accepted in planning the layout of banks in open pits. In actual mining conditions, the slopes of the banks may temporarily be somewhat steeper. The difference between the rated (design) and actual angles of slope depends on the physical properties of the rocks, the height of the bank, weathering conditions and, especially, the time factor.

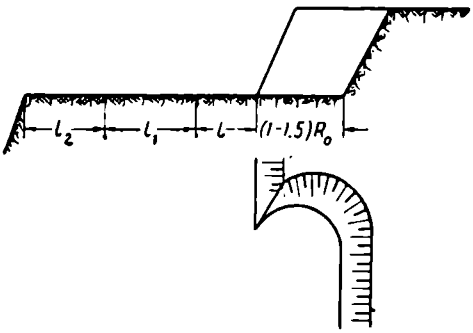



Fig. 415. Bench width in mining soft ground

The *width of the bench* (working berm of the bank) depends on the method of rock excavation.

In excavating soft rocks, the shovel digs the ground directly, and the width of its *run* is usually assumed to equal digging radius $R_0$ in operation level *1*, or within the range of (1 to 1.5) $R_0$.

The width of the bench may then be calculated as follows (Fig. 415):

$$B = (1 \text{ to } 1.5) R_0 + l + l_1 + l_2$$

where $l$ is the distance from the edge of the bank to the railway track (usually 3-4 metres);

$l_1$ — width of the railway bed (1.75 metres for narrow-gauge and 3 metres for standard-gauge tracks);

$l_2$ —margin to provide for developed reserves to be worked in the underlying bench.

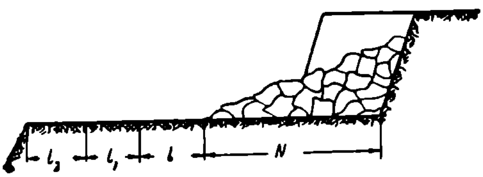



Fig. 416. Bench width in mining hard ledge rock

Hard ledge rocks are blasted and the shovel loads the material from the resultant "broken-rock pile" (Fig. 416). If the latter's width is taken as $N$, the width of the bench will then be

$$B = N + l + l_1 + l_2.$$

When the mining of banks approaches the contour lines or boundary of the pit, the slopes of the banks are not driven into the same plane. To lend stability to the edges, the width of the benches is reduced to the minimum required by safety regulations (the so-called "protective berm"). Hence the *general slope* of the

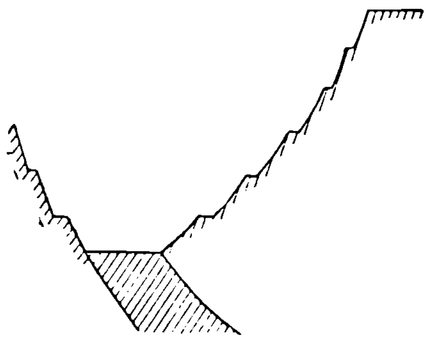


*Fig. 417.* Profile lending stability to the bank edge

pit's *edge*, determined by the angle formed by the horizontal plane and the straight line, drawn normally to the bottom outline of the pit and to the top contour intersecting it, depends not only on the bank slopes but also on the presence of protective berms. In pits of considerable depth, the general slope of the edges should be gradually flattened out (Fig. 417; pertinent figures are given in Table 19).

## 5. Volume Weights (Density), Coefficients of Expansion and Angles of Repose of Some Rocks. Angles of Bank Edges

Below are data helpful in estimating the volumes and tonnages of rocks during the planning of open-cut operations.

The *coefficients of rock expansion* following breakage depend on the condition of loose ground. The volume of rock increases maximally during its excavation. If the rock is then transported in railway cars, its volume tends somewhat to decrease again, particularly in large-capacity cars. After the disposal of the rock in dumps, it again compacts. Compaction and rain cause freshly stacked dumps to shrink or contract—in the case of loose ground shrinkage comes to 5-10 per cent and in that of lumpy rocks to 10-15 per cent. The ultimate *residual* expansion of rocks will be even less.

*Table 16*

**Volume Weight of Rocks in Place**

| Rocks | Weight in tons per cubic metre |
|---|---|
| Wet sand | 1.95 |
| Dry sand | 1.6 |
| Wet gravel | 2.0 |
| Dry gravel | 1.8 |
| River silt (sludge) | 1.8 |
| Sod (overburden) | 0.8 |
| Clay sod (overburden) | 1.2 |
| Semidry, loose clay | 1.2 |
| Wet (soaked) clay | 1.9 |
| Dense, viscid clay | 2.1 |

*Continued*

| Rocks | Weight in tons per cubic metre |
|---|---|
| Sandstones (depending on density) | 1.8-2.5 |
| Quartzites | 2.5-2.8 |
| Shales | 2.3-2.6 |
| Limestones (depending on density) | 1.5-2.7 |
| Marble | 2.7-2.8 |
| Marl | 2.3-2.5 |
| Dolomite | 2.3-2.9 |
| Gypsum | 1.9-2.6 |
| Rock Salt | 2.2-2.4 |
| Hard coal | 1.2-1.4 |
| Anthracite | 1.3-1.5 |
| Brown coal (lignite, depending on ash content) | 1.15-1.3 |
| Crystalline rocks | 2.6-2.9 |

The above is illustrated by Table 17, which refers to the typical rocks of a lignite deposit worked by a chain-and-bucket excavator.

*Table 17*

**Coefficients of Rock Expansion in a Lignite Deposit**

| Types of rock | Unit weight per 1 cu m in place, tons | Coefficients of expansion | | |
|---|---|---|---|---|
| | | in the bucket of the excavator | in railway car | in dump |
| Sand and fine pebbles | 1.7 | 1.2 | 1.15 | 1.1 |
| Loam | 1.8 | 1.8 | 1.3 | 1.15 |
| Clay | 1.9 | 1.9 | 1.5 | 1.25 |
| Brown coal (lignite) | 1.15 | — | 1.4 | — |

The *angles of repose* for loose or lumpy bodies are included in Table 18.

The general *edge slopes* of the pit banks may be taken as given in Table 19.

The figures listed in Table 19 should be regarded as rough, requiring closer and more detailed consideration in each individual case, depending on the bedding arrangement, jointing, degree of stability and sequence of rock occurrence, etc. For very deep open pits with long service-life, it is suggested to cut these figures by 3-5°.

*Table 18*

**Angles of Repose for Rocks**

| Types of rock | Angles of repose, in degrees |
|---|---|
| Pure loose sand | 32-34 |
| Loose sand with clay | 37 |
| Wet sand | 22 |
| Pure loose gravel | 37 |
| Loose gravel with clay | 37 |
| Dry loose clay | 37 |
| Solid clay in place | 40-45 |
| Moist clay | 20-25 |
| Wet clay | 16 |
| Lumpy stone rocks (average) | 38 |
| Coal | 34-40 |
| Various ores | 38-42 |

*Table 19*

**Angles of Steady Bank Slopes for Open Pits**

| Types of rock | Slope, in degrees |
|---|---|
| Soft clay ground | 25-35 |
| Heavy (compact) clay ground | 30-40 |
| Hard clay shale, sandstone and limestone | 40-45 |
| Hard sandstone, hard limestone, dolomites, igneous rocks exposed to weathering | 40-50 |
| Very hard sandstone, limestone, dolomites, metamorphic and igneous rocks | 50-60 |
| Quartzites, very hard igneous and metamorphic rocks | 60-70 |

CHAPTER XXVI

# EQUIPMENT AND LAYOUTS OF OPEN PITS

## 1. Basic Types of Open-Work Mechanisation

In modern open pits, which generally produce large tonnages and in which considerable and sometimes even huge masses of the mineral and rock are excavated and hauled, the basic operations should all be *thoroughly mechanised.*

We have already seen that mine machines and transport equipment in open pits and quarries may be large in size, since space there is not restricted as it is in underground workings.

Rock and useful mineral in bench faces are usually excavated and loaded into transport vehicles by *earth-digging machines.* The latter excavate only soft and medium-hard rocks. Hard rocks, on the other hand, are preliminarily *blasted out* of the solid mass and broken into pieces fit to handle and load, and in this case earth-digging equipment is used chiefly for loading loose "muck piles" into railway cars or other vehicles.

Earth-digging equipment includes *excavators* (power shovels and multi-bucket excavators). *Scrapers* and *crawler tractor bulldozers* not only excavate ground, but also remove or haul it over certain distances. Hence it is more fitting to designate them as *earth-digging* and *moving* machines. Placers, mainly the gold ones, are mostly worked by *dredgers.* In certain conditions rocks and mineral are extracted and conveyed by water (open-cut *hydraulicking*). Special machines and devices are used in mining *monoliths* or stone blocks for construction and sculptural purposes.

Overburden is disposed of in *waste dumps* or *spoil banks* directly by earth-digging machines or transported from working faces to spoil banks in railway cars or hydraulically. The basic operative processes at the spoil banks should also be completely mechanised.

Given below are a few typical illustrations showing mechanisation patterns in modern open pits (Figs 418 and 419).

*1.* Stripping is done by a large-capacity power shovel, which conveys the overburden to a waste dump. The mineral is excavated by a power shovel of a smaller size, which then loads it into railway cars.

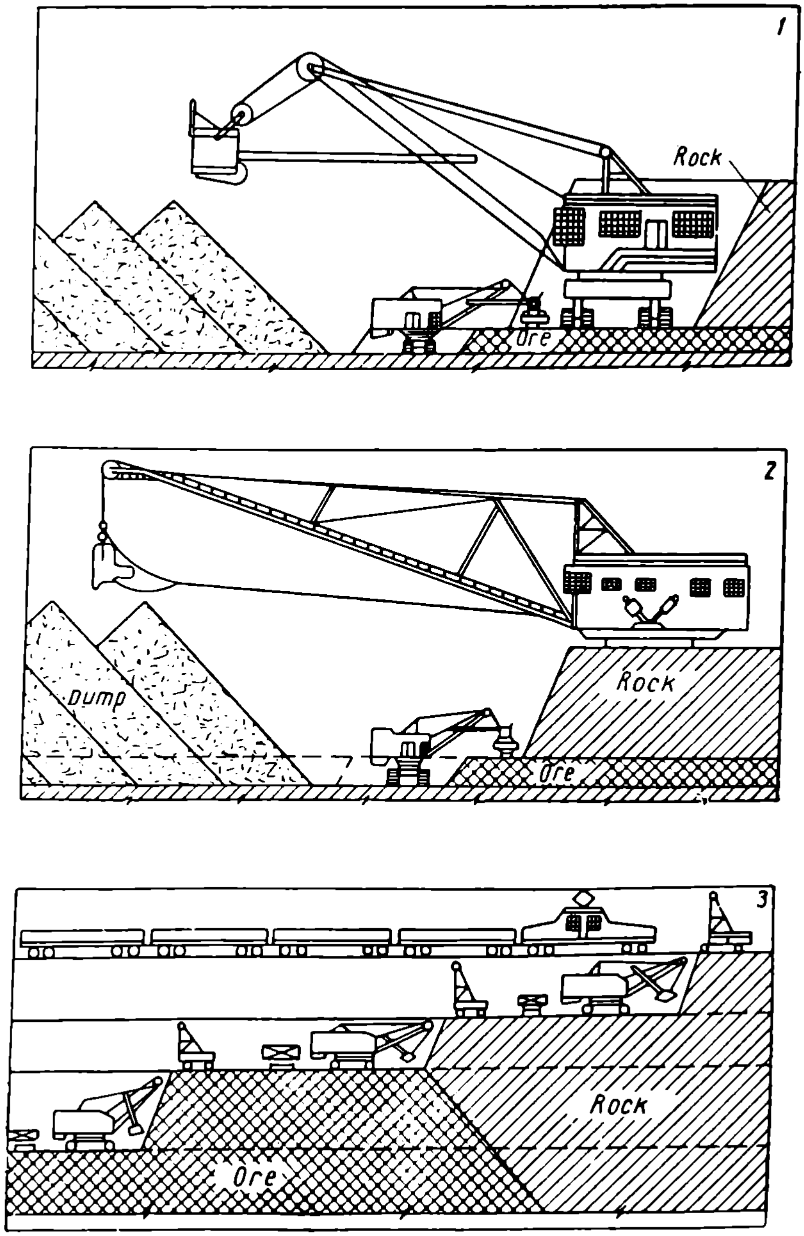



*Fig. 418.* Basic layout patterns of mechanised open pits worked by power shovels

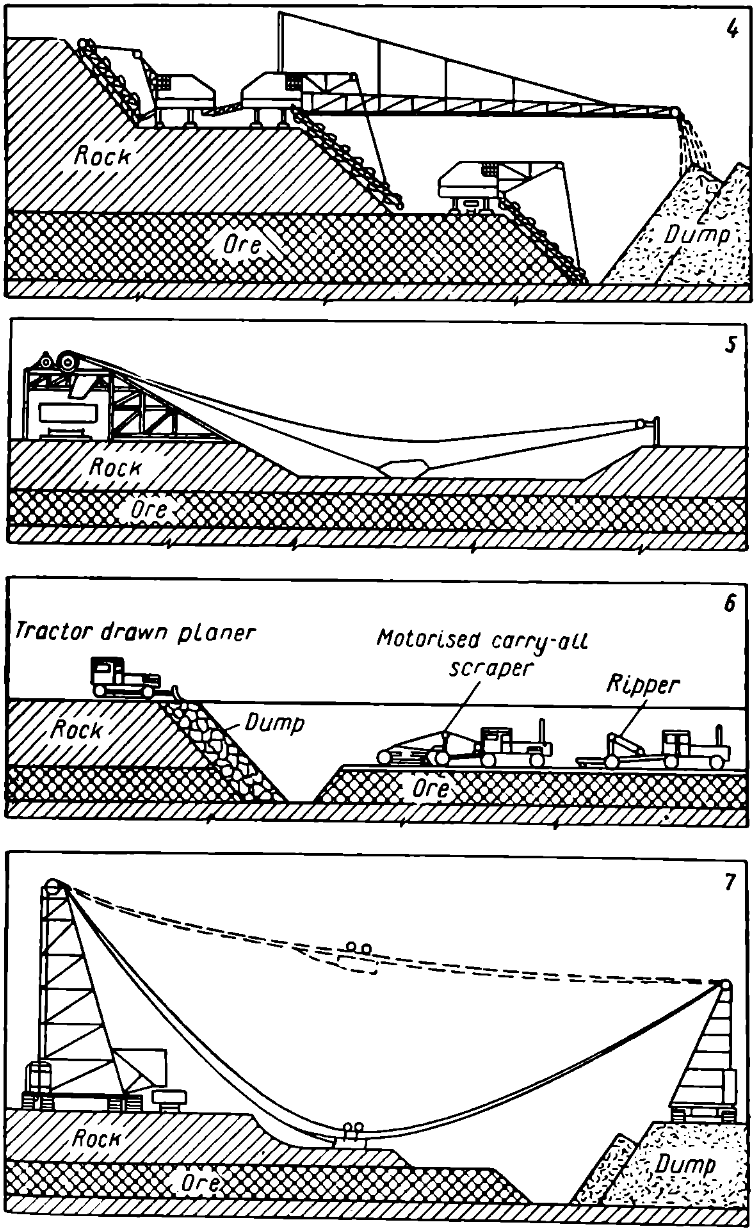



*Fig. 419.* Basic layout patterns of mechanised open pits worked by multi-bucket excavators and scraper units

2. The same pattern, but mining and overcasting are done by a dragline.

3. If the overburden at the bank faces is made up of hard rock and strong mineral, it is blasted, and for this blast-holes are made by drilling machines. The shot rocks and useful mineral are loaded by power shovels into railway cars and these go either to a spoil bank or departure stations.

4. The overburden covering the mineral is stripped by multi-bucket excavators and is disposed of and distributed in spoil banks by special long stacking conveyers.

A layout similar to the preceding one, but the overburden ground is conveyed to waste dumps by a mobile or conveyer bridge (see Fig. 446 below).

5. Mining by cable or drag scrapers.

6. Stripping and extraction of the mineral by tractor scrapers with the use of rippers and bulldozers.

7. Extraction involving the use of a tower or slackline cableway excavator.

Layouts of open-cut work involving the use of hydraulicking are dealt with in Section 10 of this chapter and with dredging in Section 11.

Let us now pass on to a brief description of the equipment employed in modern mechanised pits.

## 2. Drilling and Blasting in Open Pits

The excavation of rocks in pit banks by the blasting method usually requires vertical boreholes. In certain circumstances, however, ordinary holes of smaller diameter and coyote or tunnel charges are also employed. The pin-point method of blasting is sometimes resorted to in driving trenches.

Open-cut blasting is done chiefly with blast-holes. Their arrangement in single and double rows in the bank is illustrated by Fig. 420. In the latter case they are located in a staggered order. They

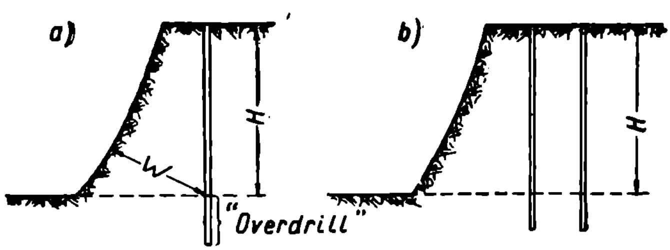



*Fig. 420.* **Location of blast-holes in a bank:**
*a*—single row; *b*—double row

have a diameter of 150-300 mm and their depth should somewhat exceed the height of the bench. In other words, the hole is somewhat "overdrilled".

N. Melnikov suggests the following standards for calculating the arrangement of blast-holes. $W$ designates the burden, that is, perpendicular distance from the centre of the charge to the bank slope. The numerical value of $W$ may be estimated as follows:

| Height of bank $H$, in metres | Burden $W$, in metres |
|---|---|
| 4.5-7.6 | $0.62H + 0.33$ |
| 7.6-18.2 | $0.24H + 3.6$ |
| 18.2 and above | $0.1H + 6.1$ |

The distance between the rows of blast-holes (given double-row holes) is $(0.5\text{-}1.0)W$, and between individual blast-holes in a row $(0.6.\text{-}1.5)W$. The charges for the blast-holes and their diameter are chosen so as to make the consumption of explosives conform with their unit standard consumption (kg/m$^3$) set for any given type of rock (Table 20).

*Table 20*

| Rock characteristics | Explosive consumption in kg per cu m |
|---|---|
| Highly strong, viscid (sticky) and dense quartzite . . | 1.3-1.5 |
| Highly strong and dense basalt, diabase and diorite . . | 1.2 |
| Highly strong granite, porphyrite and quartzite . . . . | 1.1 |
| Gneiss, granite, porphyrite, syenite, amphibolite . . . | 1.0-0.9 |
| Extremely hard limestone, sandstone and conglomerates | 0.8 |
| Very hard siderite, magnesite, sandstone . . . . . . . | 0.7 |
| Very hard shales, marble, dolomite, limestone and magnesite . . . . . . . . . . . . . . . . . . . . . . . | 0.5 |
| Sandstone and limestone . . . . . . . . . . . . . . . | 0.4 |
| Sandy shale, bedded sandstone . . . . . . . . . . . . | 0.3 |
| Hard clay shale, sandstone, gypsum . . . . . . . . . | 0.2 |
| Hard coal . . . . . . . . . . . . . . . . . . . . . . | 0.1 |

The data on the consumption of explosives in breaking rocks with the aid of blast-holes, listed in Table 20, are rough, for petrographic characteristics do not allow full determination of the blastability of ground. Also of importance here are its structure, jointing, the nature of bedding, etc.

The explosives now used most widely in open-pit work are the ammonites.

Holes and blast-holes in open pits and quarries may be drilled by the *rotary* or *percussion* method.

Vertical blast-holes in strong rocks are made by special churn-drill equipment, for instance, the БУ-2 drills (Fig. 421). This drill outfit comprises the following principal parts: frame *1*, platform *2*, mast *3*, main drive shaft *4*, spudding sheave with crankarm *5*, tool-string hoist or winch *6*, bailer hoist *7*, underframe with caterpillar tread *8*, drilling tool dismantling mechanism *9*, drill controls *10*, power plant *11*. The drill is designed for boring blast-holes up to 300 metres deep and 300 mm in diameter. To make it easier to move it around, the drill is fitted with a caterpillar tread. The operating weight of the tool string ranges from 0.5 to 1.4 tons. The drilling speed depends largely on the hardness of rocks. The performance efficiency of the churn drill per shift in the coal quarries of the Urals is about 10-20 metres, at the Magnitogorsk pit—from 12 to 15 metres and at the Ural asbestos pit—around 12 metres.

Blast-holes in softer ground are bored by the rotary drilling method with machines capable of drilling both vertical and horizontal holes. *Rotary* drilling outfits (Fig. 422) are being used on an ever-increasing scale in open pits.

Fig. 422 depicts a ПБС-110 walking type machine, manufactured by the Karpinsk Machine-Building Plant. It is intended for the rotary drilling in coal and soft ground of vertical blast-holes with a diameter of 110-125 mm and a depth of 25 metres. The outfit is furnished with electric motor *1* and a reduction gear, mounted together with hoist *2* on platform *3*. It is set up on skids or runners *4* and that makes it possible to move from place to place on the basis of the "walking" principle. Boring section *6* may be raised or lowered by rope *5* passing over sheave *7*. The per shift efficiency of the unit is up to 40-50 metres when operating in rocks and up to 140 metres in coal.

The drilling of holes a few metres deep and with a diameter usually of 30-60 mm ("shallow-hole method") is subsidiary in nature. Small and shallow holes are quite common in small open pits.

Depending on the hardness of ground, the holes are drilled mainly by pneumatic hammers of the plug or piston type, by tripod or wagon-mounted drills, or electric augers. When it is necessary to increase the explosive charge, the hole is sprung or chambered. That means its bottom portion is preliminarily enlarged by the explosion of a small charge. The small-hole method of shooting is more labour-consuming than that of big blast-holes and is therefore seldom practised.

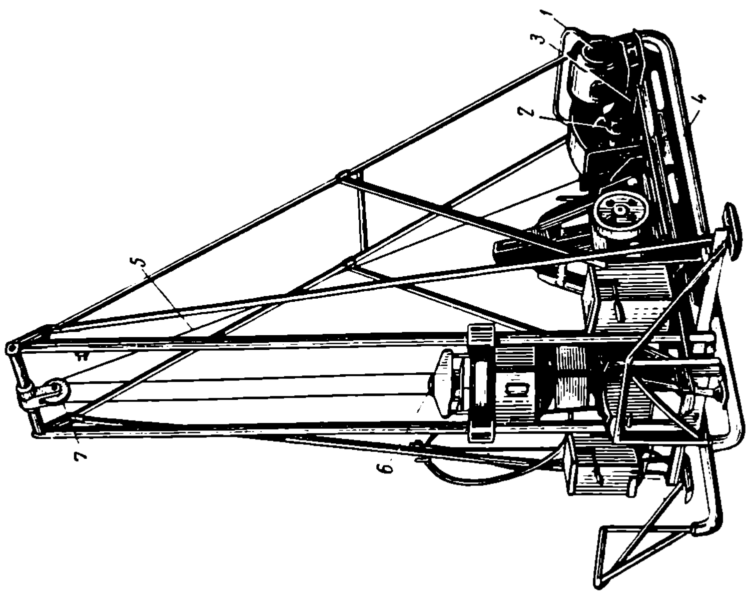


*Fig. 422.* Mobile rotary drill unit

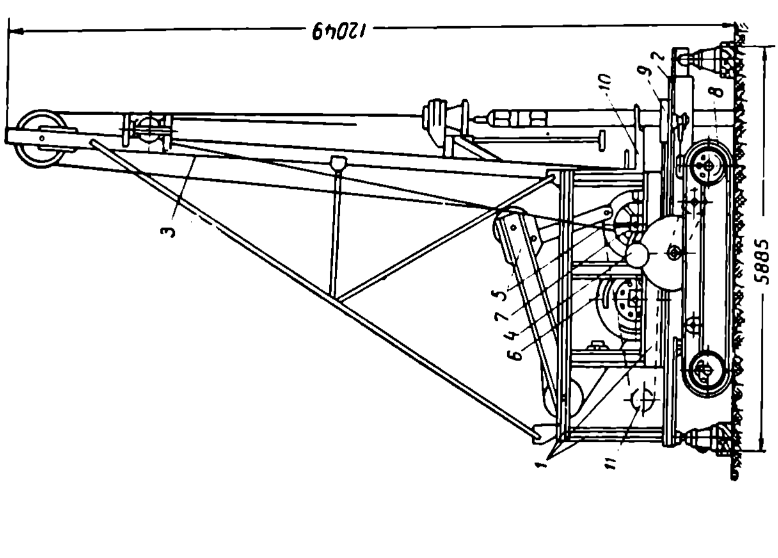



*Fig. 421.* Churn drill

Mining of rocks in quarries and pits by the *coyote* or *gopher hole blasting* is effected in two different ways—by *driving adits* and *digging holes* with powder crosscuts. The first method involves driving a small adit about 10-15 metres long into the face of the quarry at the level of the floor. From this crosscuts are made in both directions and together with the adit they form a figure resembling T. Into these workings large quantities of explosives (from several hundreds to thousands of kilogrammes) are placed at definite intervals. The explosion blasts huge amounts of rock. In the case of the second method the chamber for the explosive charge is enlarged at the bottom of a vertical digging hole. The method of coyote blasting in open-cut work is applied seldom and chiefly when local conditions make the drilling of blast-holes inconvenient or impossible.

When ground is excavated by the coyote and gopher hole blasting method, the resultant pieces or lumps may be excessive in size (oversize) and unsuitable for the transport and loading equipment available at the open pit. These lumps should be broken up and that is done by various methods, depending on their size and the hardness of rocks. If the rocks are soft, hammer piston drills weighing about 10 kg are employed. Oversize hard rocks are blasted. The explosive charges are placed into the holes drilled in rock lumps or directly upon their surface (slab or dobie charges). The latter method is the simplest, but less economical because it takes a lot of explosives and also inconvenient because it scatters the fragments over a wide area.

### 3. Mining by Excavators

At present it is *excavators* that are employed most in mining both barren rocks and the useful mineral in open pits. When the ground is soft and loose, they dig it directly at the bank face and load it into cars or other vehicles. In hard and medium-hard rocks, whose excavation requires preliminary blasting, the use of excavators is limited to loading the broken material into railway cars. In some cases, excavators (power shovels), especially big ones, are employed for overcasting the mined material directly to the spoil banks (see below).

Excavators used in open-pit work are divided into two main categories:

1. single-bucket,
2. multi-bucket.

The most important types of single-bucket excavators are the power shovel and dragline, depending on the way the bucket is connected with the driving mechanisms.

Multi-bucket excavators are sub-divided into classes of continuous-bucket or chain-and-bucket, rotary bucket and chain-and-

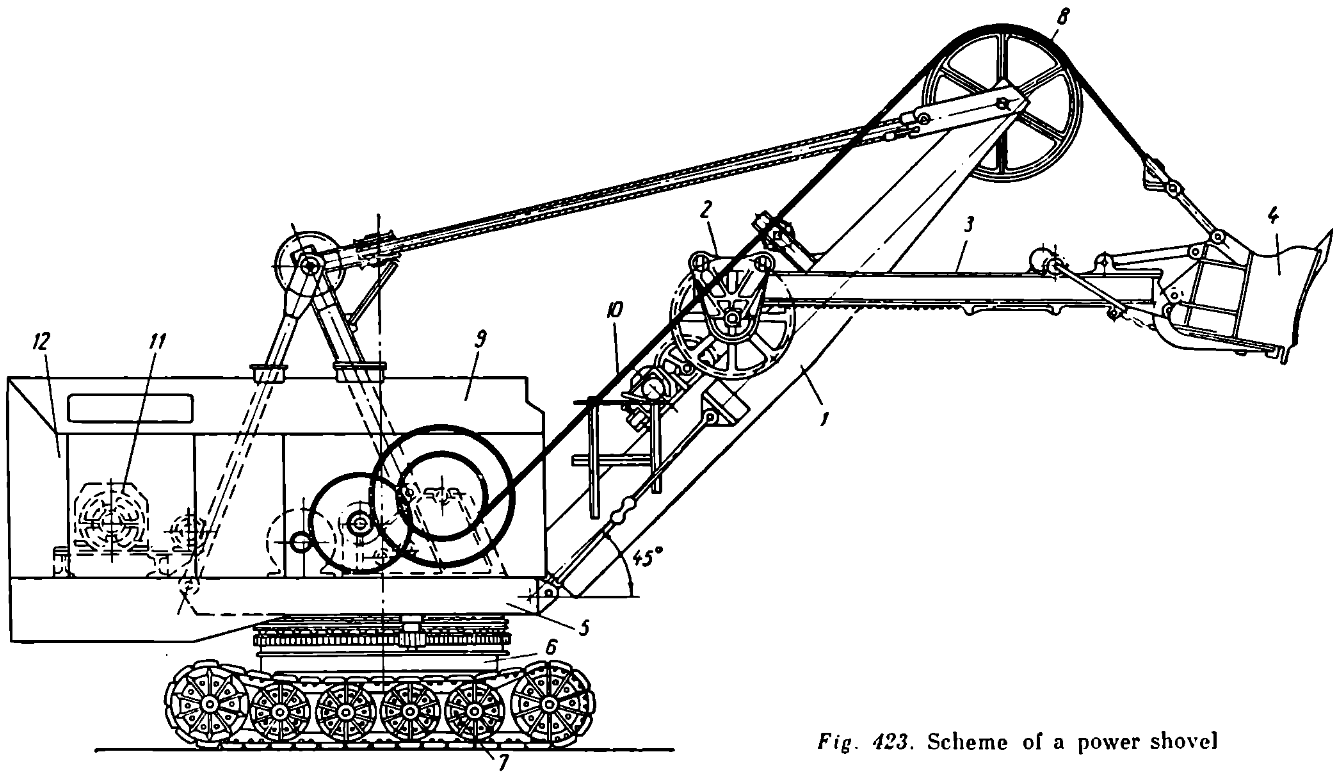



*Fig. 423.* Scheme of a power shovel

tooth machines, according to the design of their operating (working) mechanisms.

1. The *power shovel* is crawler-mounted (Fig. 423). Dipper *4*, fixed on handle or stick *3*, may perform two motions with respect to boom *1*: 1) slide along the geometrical axis of the handle which, for this purpose, is fitted with a rack driven by special crowding mechanism *2* (digging motion) and 2) move circularly in the vertical plane with the aid of cables *10*, passing over pulleys *8* at the end of the boom and wound onto drum *9* of the hoisting engine (crowding motion). These two motions ensure excavation of the material. In addition to this, the dipper can swing on turntable *5* in the horizontal plane, together with the boom and the whole of shovel body *12*, thus taking the excavated material away from the bank face and unloading it into transport vehicles (or, in casting-over operations, disposing of it directly into a spoil bank). The shovel is a self-propelling unit moving with the aid of caterpillars *7* supporting the underframe of the machine *6*. These motions of the dipper and the shovel on the whole are ensured by electric motors *11* and other mechanisms and gears in the body of the shovel. Electric power is supplied to the machine from the mains by a flexible cable wound on a drum in its body. Large-capacity power shovels have several electric motors to ensure the motions above.

According to designation and dipper capacity, power shovels may be subdivided into three distinct groups:

1) Small power shovels with dippers ranging from 0.25 to 1.5 cu m in capacity, intended for small-scale earth-moving jobs in soft ground.

*Table 21*

**Power Shovels**

| Characteristics | Models | | |
|---|---|---|---|
| | Э-10003 | СЭ-3 | ЭГЛ-15 |
| Dipper capacity, cu m | 1 | 3-5 | 10 15 25 |
| Boom length, m | 6.7 | 10.5 | 45 34 34 |
| Dipper stick (handle) length, m | 4.9 | 7.2 | 24.7 18.65 18 |
| Digging range or radius on ground level, m | 6.4 | 8.23 | 29 20.5 — |
| Maximal dumping height, m | 5.5 | 6.7 | 36 24.5 23.5 |
| Power rating of electric drive motor, kw | 80 | 250 | 1,700 |
| Number of caterpillar treads | 2 | 2 | 8 |
| Operating weight, tons | 39.14 | 165 | 1,017 |
| Estimated output, $m^3$/hr | 120-150 | 250-300 | 940 |
| Manufacturing plant | Voronezh Excavator Plant | Urals Heavy Machine-Building Works | Novo-Kramatorsk Plant |

2) Shovels of medium size with dippers holding 2-4 cu m of material; the machines of this group predominate in open-pit work.

3) Large-size shovels with dipper capacity ranging from 5 to 25 cu m. In the pits they are used mostly for stripping the overburden.

In open pits the power shovel is capable of digging any type of ground.

Soviet engineering plants now build various types of power shovels. Table 21 gives brief specifications of some.

Fig. 424 depicts an ЭГЛ-15 power shovel and its dimensions in comparison with small and medium-size units.

The United States manufactures power shovels with dipper capacity of up to 35 cu m.

The *annual* output of power shovels per cu m of dipper capacity for the available stock of these machines is: in arenaceous, readily detachable ground—250,000 cu m; in argillaceous ground with fragments of hard rock—225,000 cu m; in semihard rocks (compact marl, weak and highly fissured sandstone, shales, hard lignite and other rocks partially detached from the solid mass by blasting)—175,000 cu m, in hard rocks, with preliminary shooting—150,000 cu m. The best power-shovel crews, however, far exceed these average figures. Thus, Ushakov's crew operating a 3 cu m shovel at a Vakhrushevugol Trust pit in the Urals mined 1,638,000 cu m in 1953.

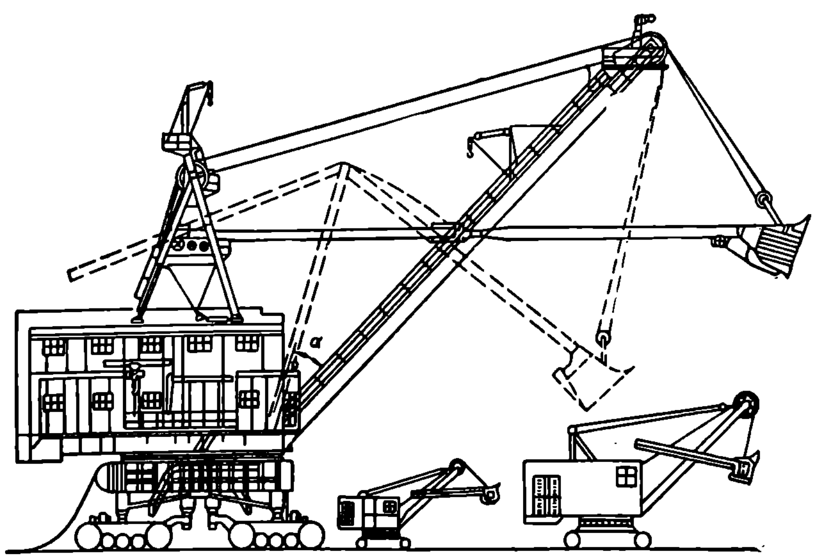



*Fig. 424.* General view of the big ЭГЛ-15 power shovel alongside small and medium-capacity shovels

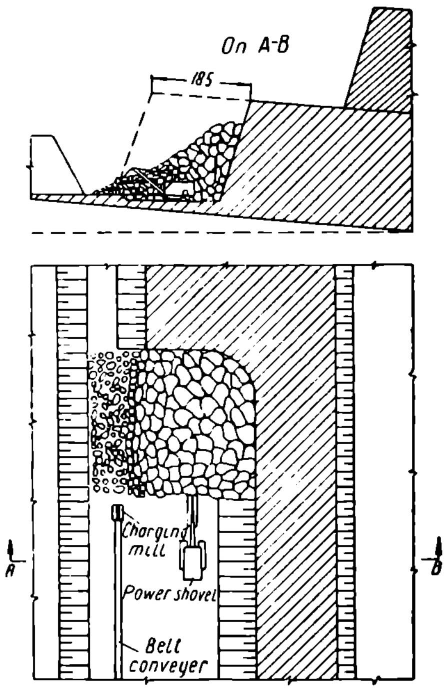



*Fig. 425.* Power shovel at a bank face

Power shovels can be operated all year round even in relatively rigorous climate, but in winter months their overall efficiency somewhat declines. For instance, in the brown coal pits of the Urals output in spring is 110 per cent of the average annual figure, in autumn—90 per cent and in winter—80 per cent.

The working position of a power shovel at the face of a coal bank at the time coal is loaded through a hopper onto a belt conveyer is shown in Fig. 425. The operation of a large power shovel engaged in stripping and *overcasting* the overburden, that is in directly disposing of it into a spoil bank, is described below (Section 7).

2. The dipper of a power shovel is fixed rigidly to its stick (handle) and boom. In *draglines*, on the other hand, it is suspended from the boom on a steel cable (Fig. 426). The design of a dragline bucket is shown in Fig. 427. It is a heavy bowl with teeth made of manganese steel attached to the digging lip below and reinforced with a bail *1* above. Load or pull line *2* is coupled to the bucket by chains *3* and hoisting line *4* by chains *5*. The bucket dumps its contents when the load line is slackened and the hoisting line is tightened. To prevent chains *5* from interfering with this operation, they are held in place by stay rod *6*. When the load line is tightened, the bucket is held in horizontal position by cable *7*, passed over pulley *8*. During the operation, the bucket is pulled by the load line along the working face and takes the ground in the same manner as the scraper. Draglines of large size may have an immense radius or reach of diggings, since the largest have booms of up to 75 metres in length. Bucket capacity ranges from one to 25 cu m and more, depending on the size of the machine.

Draglines can scoop up ground at a level considerably below that of the machine itself. Thus, the digging depth of a machine with

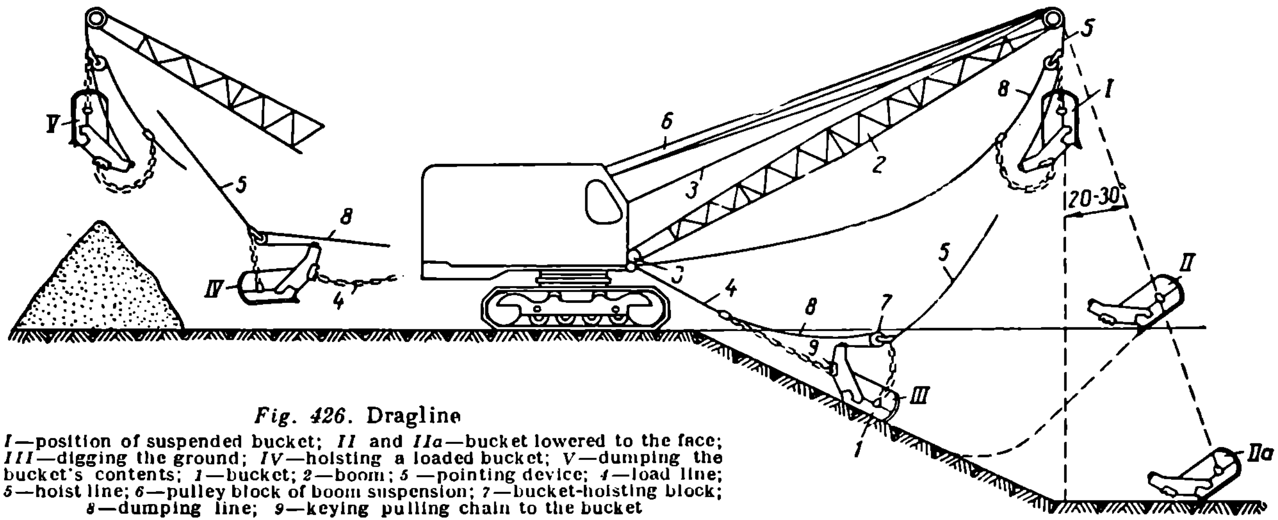



*Fig. 426.* Dragline

*I*—position of suspended bucket; *II* and *IIa*—bucket lowered to the face; *III*—digging the ground; *IV*—hoisting a loaded bucket; *V*—dumping the bucket's contents; *1*—bucket; *2*—boom; *3*—pointing device; *4*—load line; *5*—hoist line; *6*—pulley block of boom suspension; *7*—bucket-hoisting block; *8*—dumping line; *9*—keying pulling chain to the bucket a bucket capacity of 3 cu m reaches 20 metres, while that of bigger models with a bucket capacity of 10 cu m and more even 50 metres. These excavators are designed mostly for work in soft and loose ground but may also be used in handling blasted hard rock.

Draglines are employed for direct overcasting of ground without any intermediary haulage. Since the bucket is suspended from a cable line it is difficult to dump its contents at a precise spot, and so the dragline's efficiency declines when the material is loaded into rail-

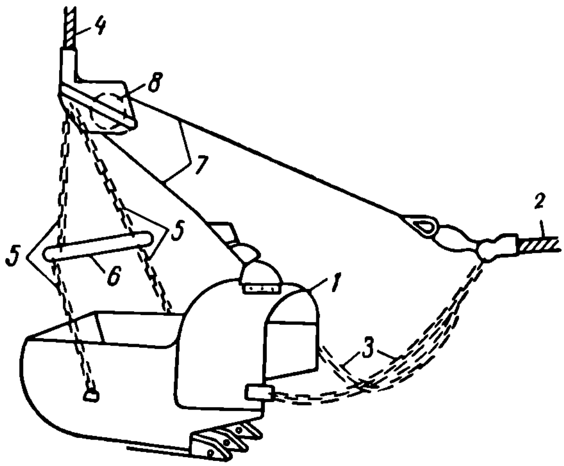



*Fig. 427.* Dragline bucket

way cars. For this reason a broad funnel-shaped hopper is sometimes placed over the car.

Like power shovels, draglines are crawler-mounted (Fig. 428). However, special *walking* draglines have been manufactured (Fig. 428) in recent years. When in operation, their body rests on a flat base whose bearing area is appreciably larger than that of caterpillar treads. Therefore, the pressure exerted by the weight of the machine on a unit of ground surface sharply decreases and a walking dragline can operate on ground into which the caterpillars would sink deeply. For "walking" the machine is provided with a unique mechanism in the shape of two side supports (shoes) resembling skis, on which it rises and moves slowly over a distance of 1-2 metres.

The large ЭШ-14/65 walking power shovel, manufactured by the Urals Heavy Machine-Building Works, is one of the most remarkable creations of Soviet technology.

Initially, the capacity of its bucket was 14 cu m and the boom extended 65 metres, and 10 cu m with the boom 75 metres long. In

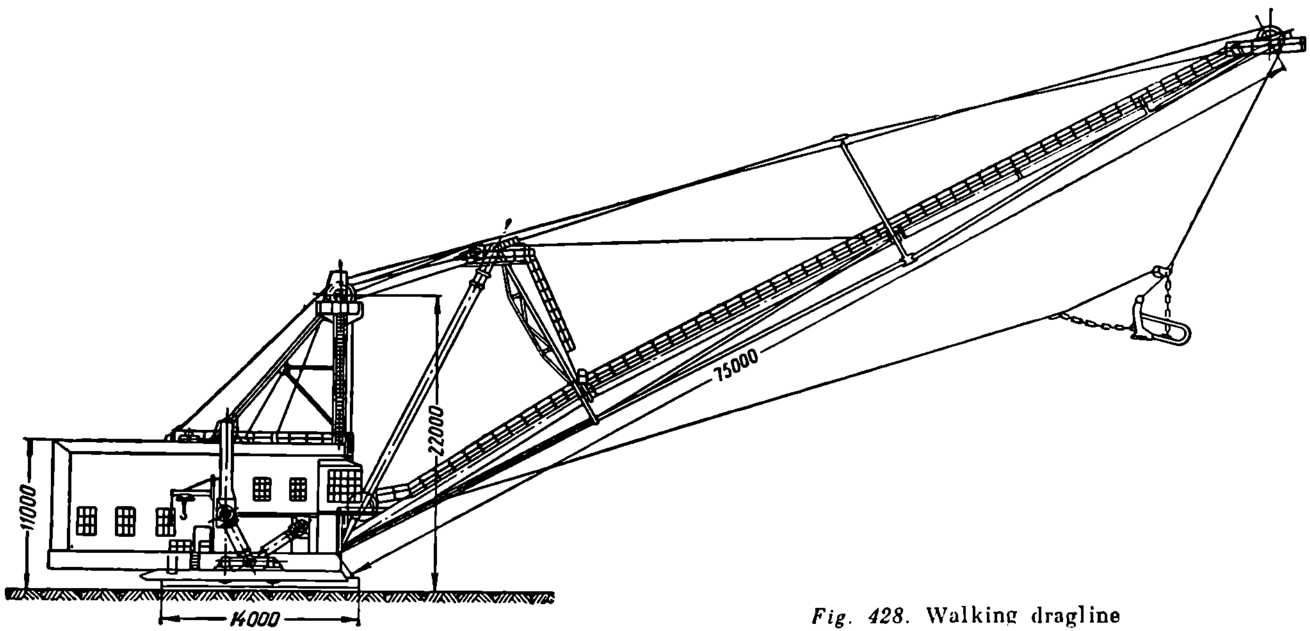



*Fig. 428.* Walking dragline

the latest models the bucket capacities have been increased to 20 and 14 cu m.

The overall weight of the machine is 1,300 tons. It can dig the ground 45-50 metres below its actual position and dump the bucket at a height of 28-32 metres above it. This makes the machine particularly suitable for overcasting work.

The walking mechanism comprises two pairs of powerful hydraulic cylinders, suspended on the sides of the machine's body, and two supporting shoes. The cylinders are fed with oil under a pressure of 175 atm. The shovel moves up to 2 metres with each step and covers a distance of 120-150 metres during an hour of continuous walking. On its route it can negotiate gradients of up to 10° and turn in different directions.

The overall rated power of the electrical motors of this shovel is about 7,000 kw.

The duration of the operation cycle is 50-70 sec. In this short space of time the shovel excavates 14 cu m of ground weighing about 20-25 tons. In 24 hours it can dig out up to 15,000 cu m.

The machine is serviced by only five or six men per shift.

Another, smaller walking power shovel, the ЭШ-4/40, has a bucket capacity of 4 cu m and a 40-metre-long boom.

In view of the rapid development of open-cut mining, the Soviet Union has built a dragline with a 125-metre-long boom and bucket capacity of 25 cu m.

The charge ratio of the dragline bucket, in comparable conditions, is somewhat lower than that of the dipper of an ordinary power shovel firmly fixed to the boom and, therefore, annual productivity per cubic metre of bucket capacity is several per cent less.

3. In open-pits it is sometimes necessary to extract relatively thin layers of mineral. For instance, in mining coal from seams of complex structure it is desirable to excavate barren rock partings selectively. This can be done by a mechanical shovel with a front skimmer attachment (Fig. 429). The skimmer differs from an ordinary power shovel by the design of its operating mechanism. Bucket *1*, with a capacity of 0.5-1 cu m, is fixed to slide block (carriage) *2*, suspended from boom *3*. In the course of actual skimming the bucket is pulled forward from the machine body by load line *4* and cuts a thin layer of ground. When the bucket reaches the tip of the boom, the latter is hoisted by cable line *5*, swings to the dumping point, and the contents of the bucket are discharged through a hinged door.

Lately a combined skimmer-planer has been designed on the basis of the СЭ-3 power shovel. It shows a marked improvement over ordinary skimmers because the horizontal boom along which the bucket slides, can be set at different levels and that greatly facilitates the extraction of beds with complex structure.

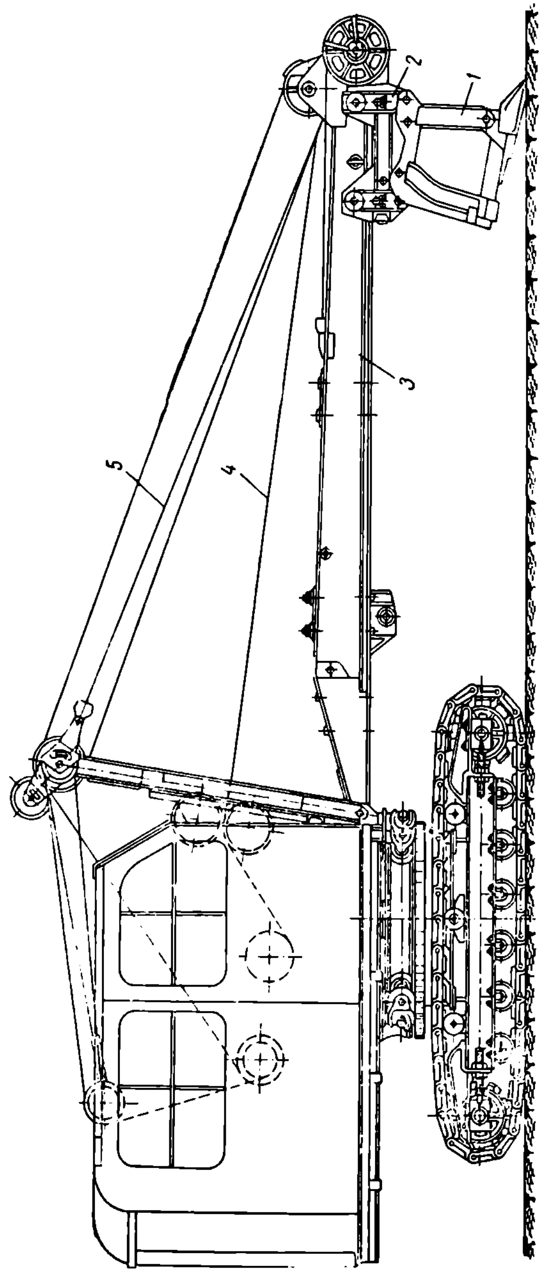



Fig 429. Self-propelling skimmer

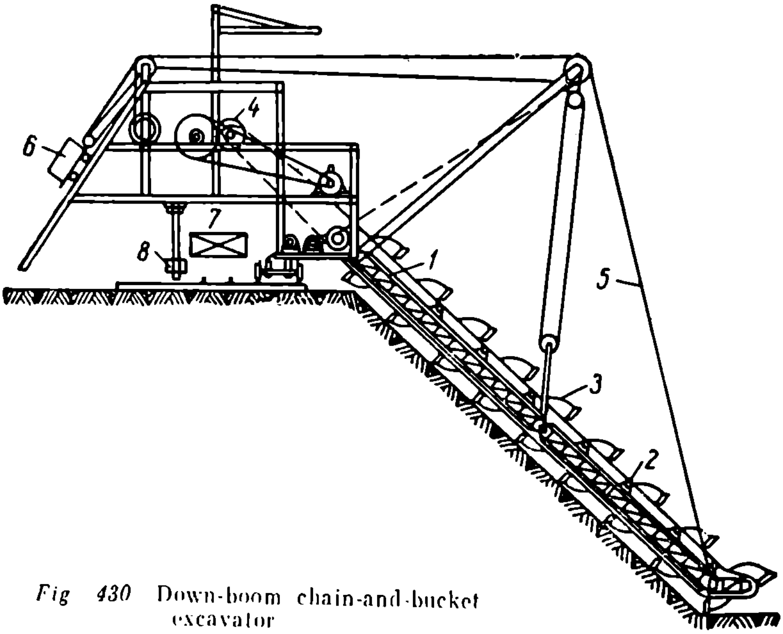



*Fig 430* Down-boom chain-and-bucket excavator

4. The principal part of a *multi-bucket* excavator (bucket-chain machine) (Fig. 430) is endless chain *2*, supported by frame *1*, with buckets *3* fixed to it. When they rise, the buckets cut and take the ground and unload it as they overturn passing over upper driving sheave or tumbler *4*. Hence, unlike the cyclic process of single-dipper power shovels, the process of digging here is a *continuous* one.

The driving gear and power plants (electric motors and more rarely steam engines) of the bucket-chain machine are set up on a special platform which can move, on wheels or caterpillars, along the face. This platform may be located on the upper berm of the bank being dug down-face (with a down-boom excavator, Fig. 430), or on the lower berm in the case of up-face digging (up-boom excavator). Some types of bucket-chain machines can be adjusted for both up- and down-face digging. Working normally, the bucket chain moves slowly (at a rate of 0.6-1.2 m/sec), while the machine itself also travels slowly along the face (at a rate of 4-12 m sec), continuously scraping the ground or the mineral off the surface of the face. With such a principle of operation, chain-bucket excavators are suitable for working loose friable or relatively soft sandy and clay ground without any inclusions of large slabs, stumps, etc. Up-face digging is somewhat less efficient than the down-face. That is why it is practised more seldom, and chiefly when the top surface of the bank is rugged.

The bucket jib hinged to the body of the machine and suspended from cables *5* or chains can be raised and lowered, and that changes the slope of the bank. To counterbalance the heavy bucket-chain jib or boom, a massive counterweight *6*, is provided on the other side of the machine. The chain bucket excavators under which loaded trains *7* can pass are called *portal* excavators. To make the body sufficiently stable, a special post or column resting on bogie or carriage *8*, rolling on a special rail, is put up to support it.

Since digging is continuous, the output of chain-bucket excavators is relatively high, although it naturally depends on the size of the machine. The capacity of the buckets ranges from 0.2 to 1.5 cu m and the estimated rated output of bucket-chain machines varies from 250 to 2,250 cu m per hour. In winter, multi-bucket chain excavators, especially the small, are less suitable than power shovels.

In building up internal spoil banks of overburden, these units can be used in combination with overburden bridges and overburden dumping machines (Section 7).

A *bucket-wheel* excavator or *land dredger* also digs continuously. Its buckets, however, are not fixed to an endless chain, but to a wheel, that is, a rotary movement is imparted to them during the operation (Fig. 431).

The body of the bucket-wheel excavator rests on underframe *1*, fitted with crawler bogies or caterpillar wheels. The body of the machine is made to turn around its vertical axis by swinging mechanisms *2*. Operating wheel *3* of the excavator has eight or six curved blades (buckets) *4*. The wheel itself is set up at the end of frame (jib) *5*, shaped like a girder truss or beam. The jib carries belt conveyer *6*, which transfers mined ground to the loading, tail part of the machine, made in the shape of a swinging cantilever or arm *10*. This cantilever is furnished with belt conveyer *9*. Rocks are dumped into a railway

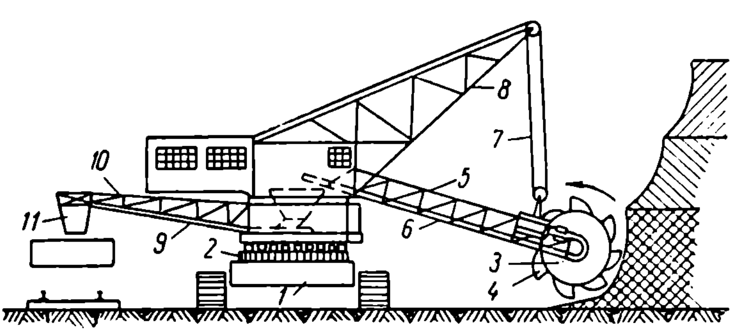



*Fig. 431.* Bucket wheel excavator

car or a truck through chute *11*. The jib can be raised and lowered together with the bucket wheel by cables *7*, suspended from boom *8*.

The excavators of this type can work only in dry loose ground. They are used mainly for selective layerwise excavation of barren rocks and minerals.

## 4. Employment of Earth-Digging and Moving Machines

The operating organ of a *scraper* is also a bucket of one or another shape. But whereas excavators are intended only for detaching the rock from its solid mass or for scooping the loose ground and loading it into vehicles, scrapers besides *haul* the obtained material over a certain distance.

The scraper bucket is moved either by a cable line from a hoist or by a tractor.

Cable or drag scraper plants are now almost out of use in opencast work.

*Tractor (wheel) scrapers* have been used in open pits in the past 10-15 years. The bucket or body of the scraper, especially designed for this purpose, is mounted on truck-type wheels (Fig. 432). The scraper is hauled by a tractor. Its bucket is filled and unloaded while it is in motion. To dump the load, the scraper bucket is tilted, or the material is pushed out by a special movable plate fitted inside the bucket. The unit is operated by the tractor driver with the aid of wire cables. To increase the cutting force and accelerate the operation of large-sized wheel scrapers, pusher-tractors are sometimes

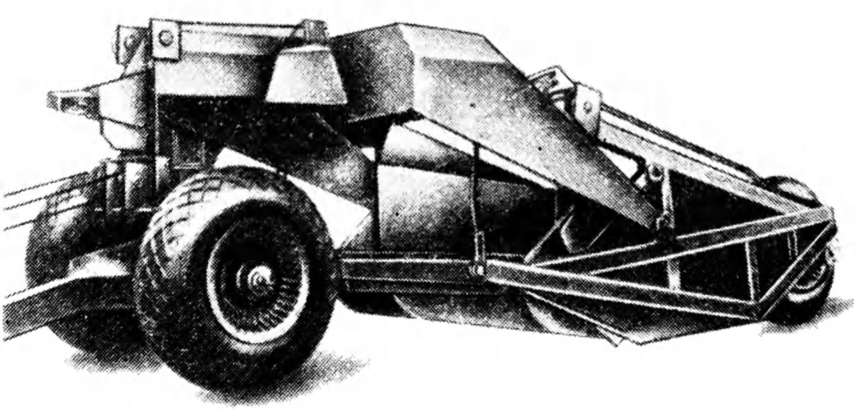


*Fig. 432.* Trailer wheel scraper

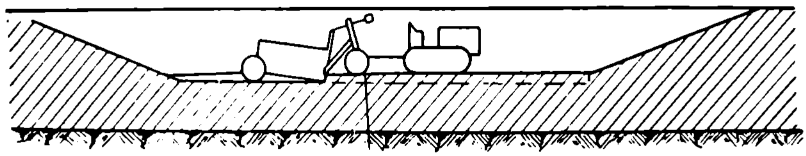



*Fig. 433.* Diagram illustrating the operation of a wheel scraper unit in an open pit

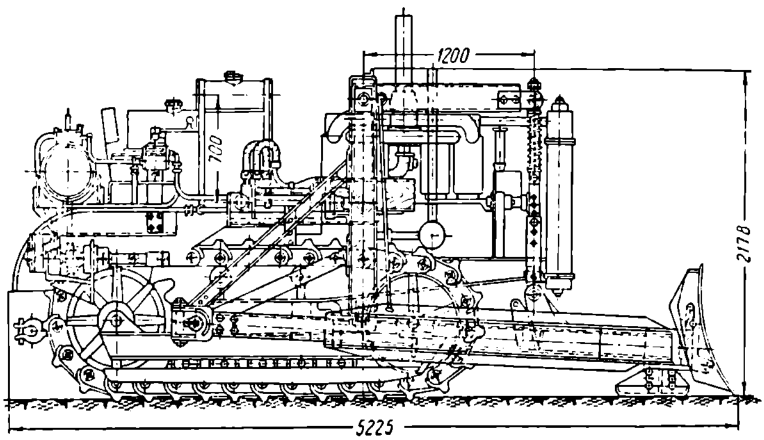



*Fig. 434.* Bulldozer

used for assisting them in loading. Fig. 433 is illustrative of the pattern followed in operating a trailer scraper unit in open-pit work.

Rapid progress has been achieved in the manufacture of scrapers. There are already models with capacities of up to 25-30 cu m. The Chelyabinsk Tractor Works puts out wheel scraper units with capacities of 6.5 (overall weight 7.2 tons), 10 and 15 cu m (weighing 14 tons). A 6.5 cu m scraper is pulled by 80 hp tractor, a 15 cu m unit requires a 140 hp tractor and a pusher-tractor.

Somewhat larger models with capacities of up to 17.5 cu m are manufactured in the U.S.A. The output of a 6 cu m unit with hauls within the range of 200 metres comes to several hundred cubic metres per shift, depending on working conditions.

Wheel scrapers are very convenient for earth moving and digging, for they move and turn easily on sinuous roadways and negotiate gradients of up to 20-25°. But they can be operated only in loose and soft ground.

As independent earth-digging and moving units, wheel scrapers can be employed for small-scale open-cut work, while in large pits their use is restricted to ancillary operations (cleaning up stripped areas; operating in sections of the pit inaccessible to power shovels; making shallow trenches pending the arrival of main excavating and transportation equipment, etc.).

Closely resembling wheel scrapers by its operating features is the bulldozer or bullgrader, in which work is performed not by the bucket, but by a ploughshare fixed in front of the tractor and pushing the ground it cuts ahead (Fig. 434).

### 5. Mining of Monoliths (Ashlar or Cut-Stone Blocks)

In mining stones for construction and sculptural purposes it is important for the blocks to have a definite shape and size and to be free from cracks and fissures when they are excavated in the working face. This facilitates their further processing and reduces the loss in the form of fines. But that cannot be achieved by blasting operations and for this reason monolith stones are mined by specially designed machines. For example, marble blocks and other building stones are excavated by a machine of a type shown in Fig. 435, by a self-propelling unit moving by rail and cutting stone blocks. The machine has been invented by A. Stolyarov.

Since monoliths may be of considerable size and weight, the quarries are equipped with hoisting and jib cranes (Fig. 436) to take blocks from the working faces to where they are loaded into vehicles. The cranes of this type consist of mast *1*, held in place by a

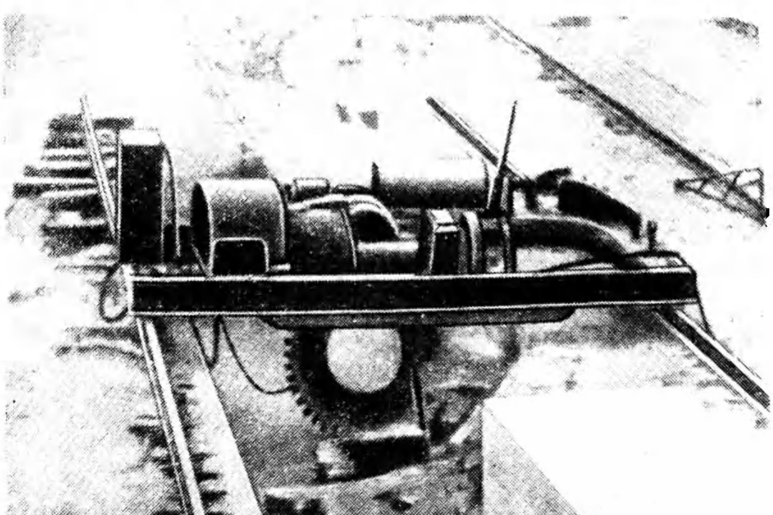


*Fig. 435.* A. Stolyarov's machine for cutting marble blocks

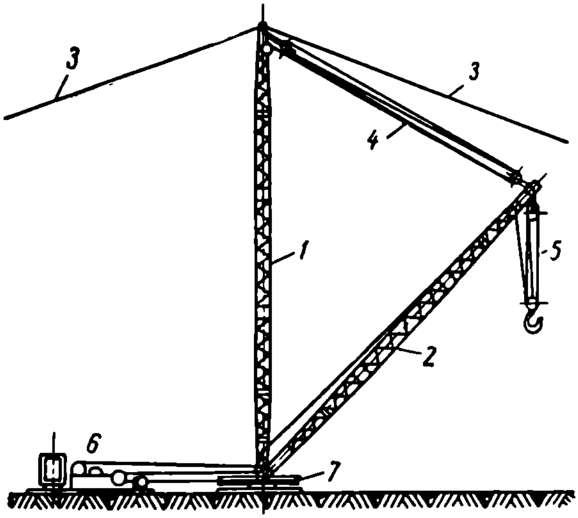



*Fig. 436.* Hoisting jib crane for handling stone blocks

number of cable support guys *3*. The stone monolith is suspended from swinging jib *2* by pulley block *5*. The angle of the jib's inclination can be changed with the aid of pulley block *4*. The jib can be swung around on turntable *7*. The jib and hoisting pulley block *5* are driven by hoists *6*.

Special methods are employed to obtain stone blocks of a particularly large size. Thus, the marble used for the interior decoration of the Ethnographical Museum in Leningrad came from the Belaya Gora deposit in the Olonets area. The job required round columns 9 metres high and 0.85 metres in diameter. To obtain marble blocks suitable for this purpose the following method was used. Because of the natural features of the structure of the occurrence and the exposure on the lakeshore of one of its sides, about 25 metres high, it was decided to detach a block of huge size—20 × 15 × 13 metres. To do this, vertical cuts 1.5 metres in width were carefully blasted out on the flanks of and all the way up the stone mass. These cuts separated the block from the parent rock. Another cut was made below the block, 4 metres high on the outer side and one metre high on the inner. To make this cut two adits were driven under the stone mass. They were gradually connected with each other and the side cuts, from the rear side of the block to the mouth of the adits. This involved leaving temporary protective marble pillars near the mouth of the adits. The size of the pillar area was estimated on the basis of the ultimate compressive strength of Belaya Gora marble (2,100 kg/cm$^2$). The top surface of the block was cleared of ground and boulders, after which three vertical blast-holes were drilled in its rear side then charged with 200 kg of black powder. To detach the block completely, numerous holes were drilled in protective pillars and charged with dynamite. The charges in holes and blast-holes were electrically fired all at once. The blasted block measured 4,200 cu m and weighed 11,600 tons. In falling, it broke into three parts, and out of these monoliths of required size were cut for the columns.

## 6. Transport Facilities in Open-Cut Mines

The vehicles used most for the transportation of the mineral and barren rock in modern mechanised open-pits are large-capacity dump cars, hauled by electric or steam locomotives, self-discharging trucks and belt conveyers.

1. *Large-capacity dump cars* should be made convenient for rapid loading by power shovels and automatic discharge. That is why they are made open. There are various ways of automatically discharging them. The most popular design is a car whose body tilts in the dumping process, turning round the long axis, and its side board rises simultaneously (Fig. 437). The body is tilted at an angle of 40-45° by air cylinders set up on the underframe of the car (Fig. 437). The body of a small-capacity dump car is tilted by hand with the aid of special levers. This, however, consumes much time. Large-capacity dump cars are mounted on two double-axle wheel bogies. The 50 cu m capacity dump car widely used in open-pit mining has the following basic characteristics: track gauge—1,524 mm (standard), capacity—22.6 cu m, dead weight—31.5 tons, width—3.15 metres, height—2.9 metres, length between couplings—12.8 metres, number of air cylinders—4.

As said above, large-capacity dump cars are used most for transportation purposes in big open pits. For small-scale open-cut work narrow-gauge mine cars of the rocker type are used (Fig. 438). They are built for a track gauge of 750 mm, in capacities of 0.75, 1 and 1.5 cu m and with load-carrying capacities of 1.5, 2 and 2.7 tons

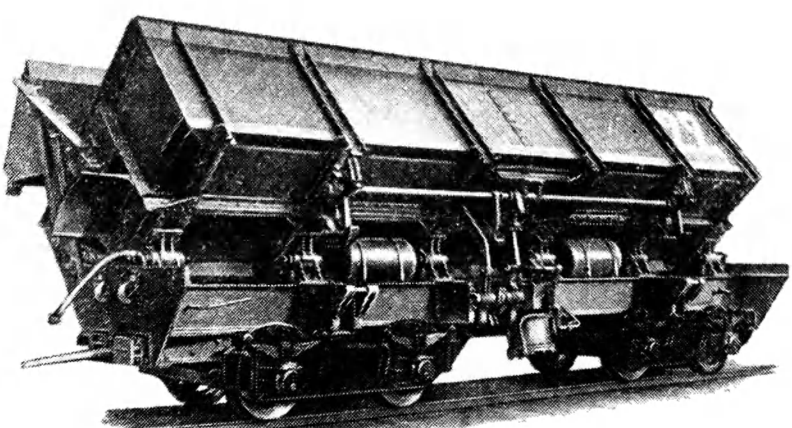


*Fig. 437.* Compressed-air dump car

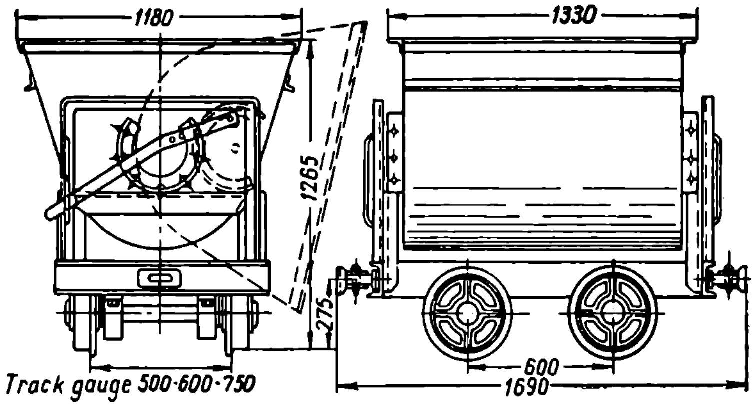



*Fig. 438.* Rocker car

respectively. Larger mine cars for narrow gauge have capacities of 2.5, 3, 5 and 6 cu m and load-carrying power of 4 and 9 tons. Small-capacity rocker cars are manufactured by many Soviet plants.

2. *Electric locomotives* are by far the most important type of traction for trains composed of large-capacity dump cars. The ones now used mostly in open-cut work are listed in Table 22.

*Table 22*

**Electric Locomotives**

| Characteristics | Models | | | |
|---|---|---|---|---|
| | 1У-КП-1 | ПЭ-150 | 13-Е-1 | — |
| Weight, tons | 80 | 150 | 150 | 44-70 |
| One-hour power rating, kw | 832 | 1,440 | 1,560 | 60-185 |
| Direct current voltage, v | 1,650 | 1,110 | 1,500 | 1,100-600 |
| Tractive effort, tons | 12 | 22.5 | 19.8 | 8-13 |
| Speed, km/h | 24.7 | 23 | 28 | 13-18 |
| Gauge, mm | 1,524 | 1,524 | 1,524 | 900 |
| Least curvature radius, metres | 40 | 80 | 80 | 60 |

The Dynamo Works also manufactures narrow-gauge electric locomotives with adhesion weight of 30 tons and tractive effort of 3.7 tons (at one-hour rating) operating on 550 v and powered by a motor of 67.5 kw.

In open pits with a nonelectrified transport system large standard-gauge *steam locomotives* may be employed, or special types of narrow-gauge locomotives, weighing 15-20 tons and with a tractive effort of about 3-5 tons.

In open-pit conditions, particularly in winter, electric traction has overwhelming advantages over steam locomotives.

Track shifting in open pits and at waste-dump sites is dealt with in Section 7 of this chapter.

3. The progress made in designing and manufacturing *automobiles* has of late led to an ever-increasing utilisation of automotive transport in open pits.

Motor trucks employed for open-cut work should be adapted for loading by power shovels and provided with automatic dumping devices. The body of the truck is made strong to withstand the impact of big rocks falling from a considerable height. For automatic discharge there are *dump-body trucks* (Fig. 439).

The body of these trucks is raised by hydraulic jacks to tilt 70°, this ensuring the complete dumping even of sticky clay ground. Basic information on dump trucks now manufactured in the U.S.S.R. is given in Table 23.

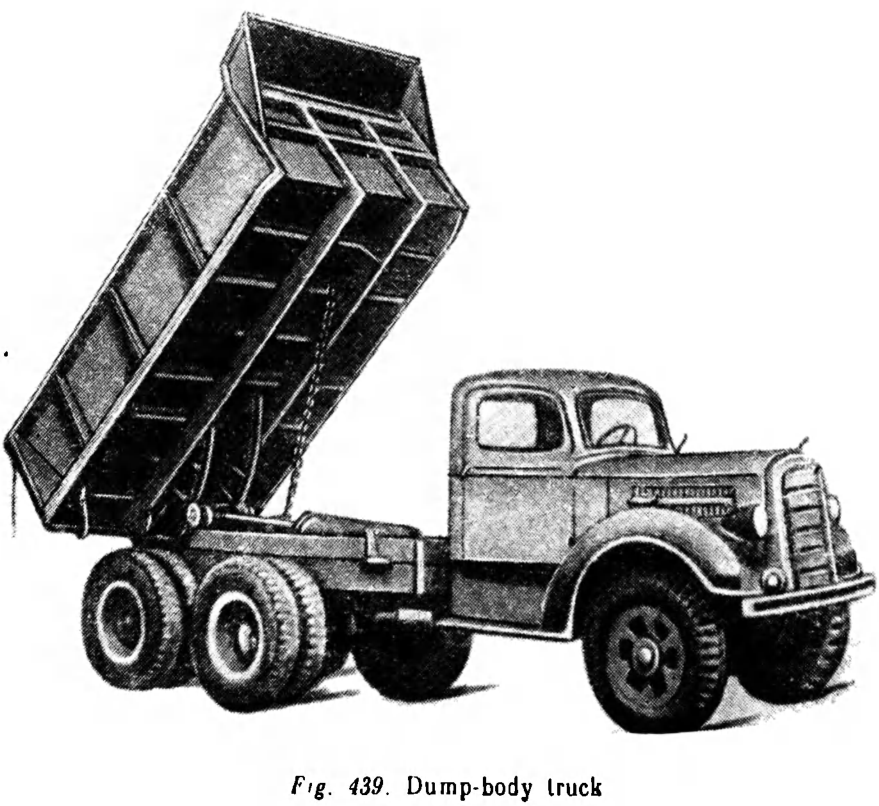


*Fig. 439.* Dump-body truck

*Table 23*

**Dump-Body Trucks**

| Characteristics | Models | | | |
|---|---|---|---|---|
| | ЗИЛ-585 | МАЗ-205 | ЯАЗ-210-Е | МАЗ-525 |
| Load-carrying capacity, tons | 3.5 | 5 | 10 | 25 |
| Motor power rating, hp | 90 | 110 | 168 | 300 |
| Type of fuel | Petrol | Diesel oil | Diesel oil | Diesel oil |
| Maximum speed, km/h | 65 | 55 | 45 | 30 |
| Manufacturing plant | Likhachov Automobile Works, Moscow | Minsk Automobile Works | Yaroslavl Automobile Works | Minsk Automobile Works |

As we see, the load-carrying capacity and motor-power rating of dump-body trucks manufactured in the Soviet Union are very high.

There are even bigger dump trucks in the United States, with a body capacity of 24 cu m, carrying load of 45 tons, and two motors of 600 hp.

Other types of automotive vehicles for open-pit work are trucks and articulated tractors with automobile wheels or caterpillars and *trailers*. Fig. 440 shows that trailers of this kind have bodies which, like dump cars, automatically discharge the material. The load-carrying capacity of such trailers may be very high (up to 70 tons in the U.S.A.)

In open-pit conditions the use of automotive transport offers a number of material advantages. For instance, curvature radii can be kept down to a minimum. Road gradients can be as high as 10-15°. The small curvature radii and steep gradients make it possible to mine small open pits, with sinuous outlines and rugged topography.

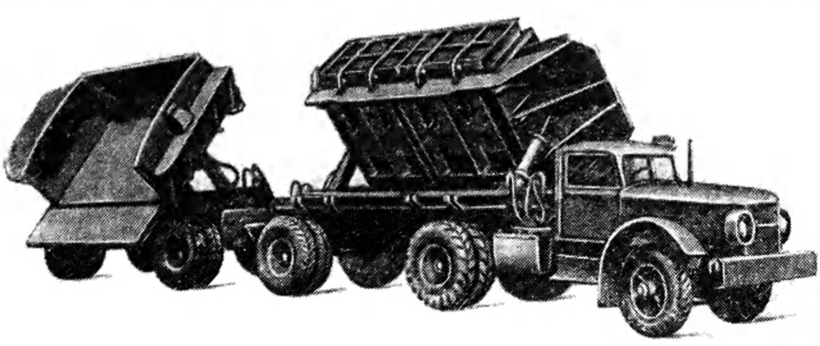


*Fig. 440.* Dump truck with trailer

*Fig. 441.* Trolley dump truck

Motor trucks and trailers can easily be moved from place to place. A dump truck can be positioned most advantageously while it is loaded by a power shovel, and that considerably raises the latter's efficiency. More, if one motor truck breaks down, that does not stop the other machines.

On the other hand, heavy dump trucks and trailers need good roads. Rainfall and severe frost sharply reduce their efficiency. Automobiles consume valuable and sometimes scarce liquid fuel.

Consequently, in open-cut mining automobiles are used to the best advantage in small pits with a short service-life and low reserves of the valuable mineral, and especially in deposits with irregular contours occurring in mountainous country. In large pits and quarries automobile transport is advantageous when they are worked at a considerable depth; in this case it is possible to avoid the immense volume of work essential to provide exits for locomotive-drawn trains. Automotive transport may successfully be employed in the initial stage of open-pit construction, when there is not enough working space for the basic equipment to operate in. Motor transport in open-cut mining is economical when the hauls are relatively short—3-5 km.

4. *Trolley dump trucks* have been used at the Boguraevsk limestone quarry (Fig. 441) since 1952. Used for this purpose are the chassis

and body of dump trucks with load-carrying capacity of 5, 10 and 25 tons, equipped with electric motors fed from trolley wires through rod current collectors. Trolley trucks have many important advantages over ordinary motor dump trucks: there is no need of liquid fuel; electric motors require less repairs than internal combustion engines and their tractive performance is better; they work efficiently in winter; there are no exhaust gases and so drivers work in better hygienic conditions.

Their manoeuvreability, on the other hand, is somewhat inferior to that of automobiles, but sufficient for open-pit work, inasmuch as it can run 4-5 metres from the trolley wire centre line. Trolley trucks require the establishment of a direct current traction substation and trolley wires.

5. *Belt conveyers* are widely used in open pits and in some other instances for the transportation of coal. The standard width of the belts is 700, 900 and 1,200 mm. The loading of the material onto the conveyer and its uniform delivery is ensured by a travelling hopper with a feeder set up over the conveyer (Fig. 442). In the case of complex seams some of the waste left in the pit, when possible, can be picked out on the conveyer. The belt incline of the conveyer should not exceed 18-20°.

To distinguish it from haulage by rail, transportation by motor trucks and conveyers is called *flexible*.

6. When little mine cars are used in working deep but small open pits and quarries, they can be brought to the surface by *inclined hoisting plants* with decks (cages) accommodating one or two cars. Hoists equipped with *skips* (see Fig. 463) are more efficient.

7. Hydraulic transport is dealt with in Section 10 of this chapter.

## 7. Waste Dumps and Equipment

1. The problem of locating and arranging waste dumps and organising and mechanising *spoil disposal* in open-cut work is extremely important, since in pits of large size waste dumps and spoil banks may assume tremendous proportions. The cost of overburden transportation may then constitute a substantial part of total mining expenditure.

The choice of the site for waste dumps should take into account the development of the pit throughout its service-life. There have been cases of no provision being made for the development of open pits in the initial stage of their operation and of using tracts of land, which could later be used as quarries, for waste dumps. Shifting these dumps to a new site entails considerable and useless expenditure. For example, it would be a mistake to arrange waste dumps at the foot wall of a steep deposit (see Fig. 410) in position $A'$, where a vertical shaft has to be sunk later. The proper site would be

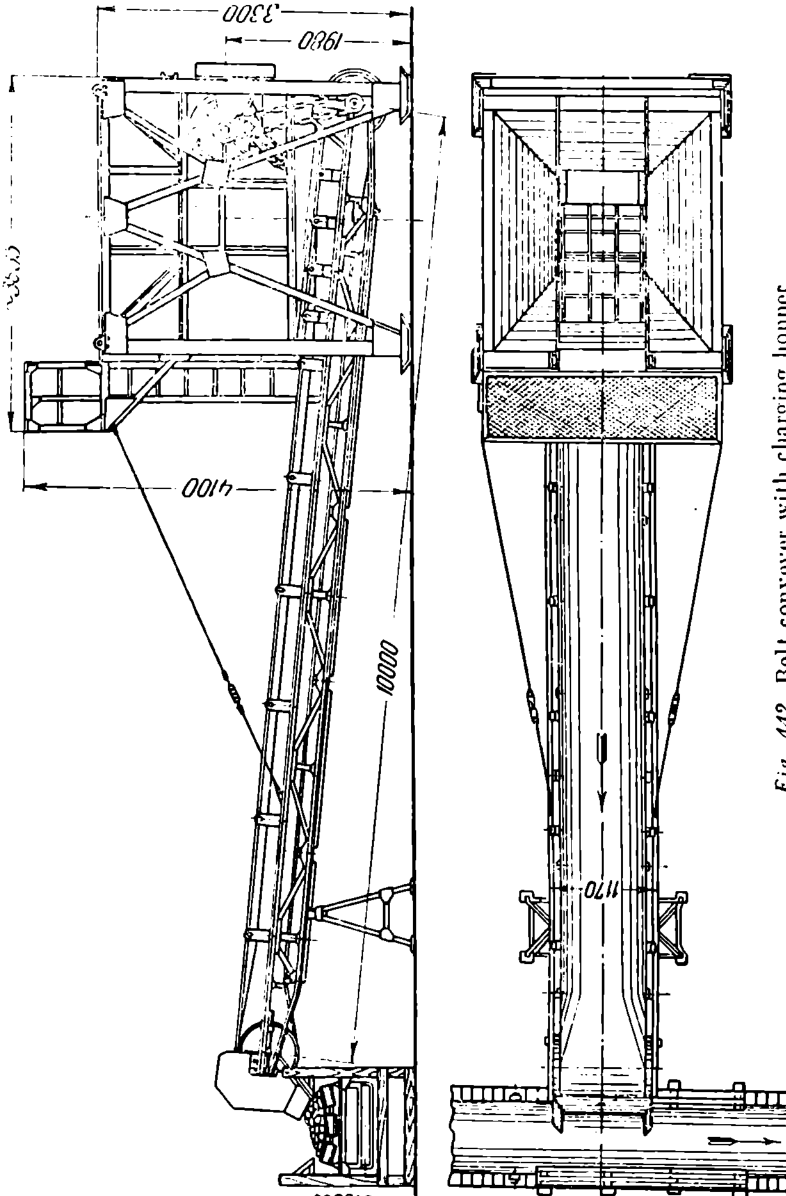



*Fig. 442.* Belt conveyer with charging hopper

position $A$. Accordingly, the appropriate place for spoil banks at the hanging wall would be $B$ and not $B'$. In selecting the site for a waste dump and laying transport ways and roads one should take account of the actual position of mine workings, disposal of the spoil and lay of the land.

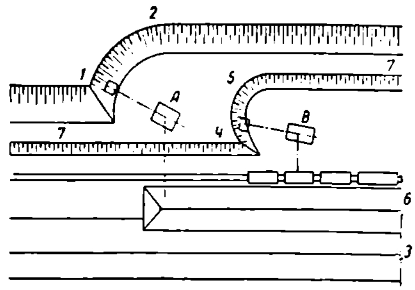


*Fig. 443.* Position of power shovels, working faces and waste dumps in overcasting operation

Waste dumps are classified into internal or inside and external or outside.

2. The *inside* dumps are those where the spoil is disposed of in the pit itself. For inside dumps there is no need of plots that could be used for agricultural purposes. Transportation of overburden to inside waste dumps is simple and cheap. It is especially so when the attitude of the deposit and the availability of a large-capacity excavator (power shovel) permit *overcasting* or course stacking, that is, direct disposal in the pit of the spoil stripped by the power shovel (see Fig. 418). Overcasting and course stacking have the enormous advantage of obviating any supplementary transportation. This method is applicable when the deposit is a horizontal or slightly sloping occurrence and the thickness of cover rocks permits the excavation of overburden in one bench. The position of power shovels and the location of working faces and spoil dumps in an overcasting operation are shown in Fig. 443. Here stripping power shovel $A$ digs the ground at face *1-2* and disposes it directly in waste dumps or spoil banks *3*. Coal (or any other mineral) is mined by coal power shovel $B$ of smaller size in working face *4-5*, and is loaded into open railway cars spotted on tracks *6*. Pass-byes with switches are arranged for the shunting of loaded cars and their replacement by empties. Berm *7*, sufficiently big for the stripping shovel to return to make full-length cut, is left between the slopes of banks, on the overburden or coal. When low coal seams are worked and the overburden is overcast directly to a spoil bank, the economically justifiable overburden ratio may be as high as 10-15 and even more.

Inside waste dumps or spoil banks can also be built up with the aid of belt conveyers mounted on long suspended cantilever trusses (Fig. 444). Such mobile overburden dumpers and spreaders are used at the Lopatinsk phosphorite quarry (Fig. 445). From a stripping chain-bucket excavator the spoil is carried to the dump by two mobile belt conveyers.

A further step in the direction of rational spoil disposal inside the pits has been made with the application of *overburden bridges*. A

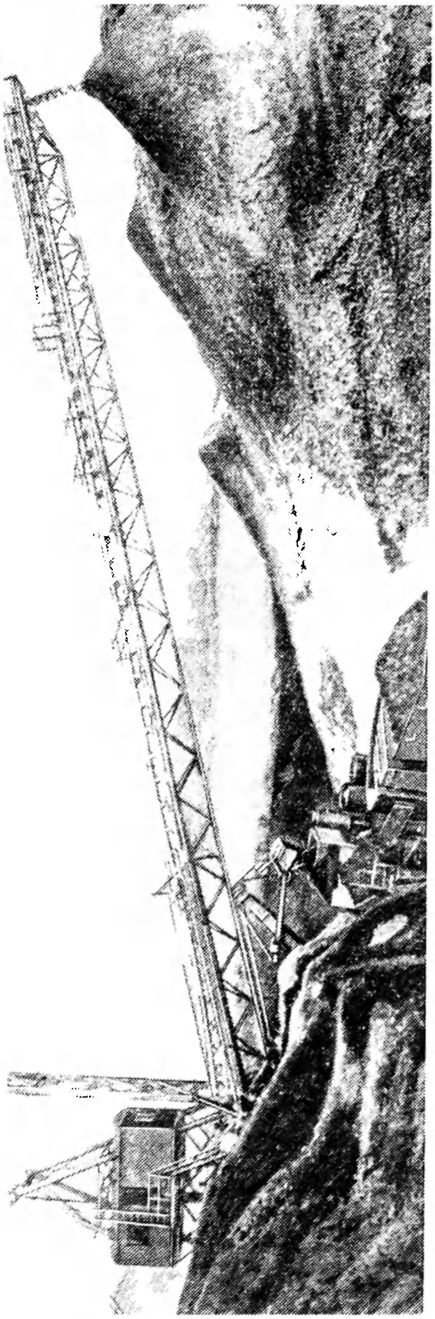


*Fig. 444.* Overburden dumper with belt conveyer

diagram of one of such arrangements is given in Fig. 446. On the right one sees a large multi-bucket excavator stripping rocks capping a brown coal deposit. The stripped overburden is fed to a belt conveyer mounted on an overburden bridge. The latter rests on two supports which move on rails together with the stripping excavator. On the excavator side the bridge has a boom in the shape of a raised cantilever, adjusted to suit the height of the spoil bank. To make waste dumps uniform, the spoil can be discharged at three different points. The excavation and transportation of the overburden thus proceed *without interruption*. This not only makes it possible to use bridges of enormous size but also greatly to increase their capacity. The overall length of these bridges (together with the boom) reaches 300 metres and their height from the ground to the lower girdle—50 metres. They are capable of handling up to 2,000 cu m of spoil per hour. In spite of considerable initial outlays, the high efficiency of the machine and small service crew required make stripping costs very low. Overburden bridges, however, can be used only in certain conditions: when the occurrence is horizontal and regular, the country

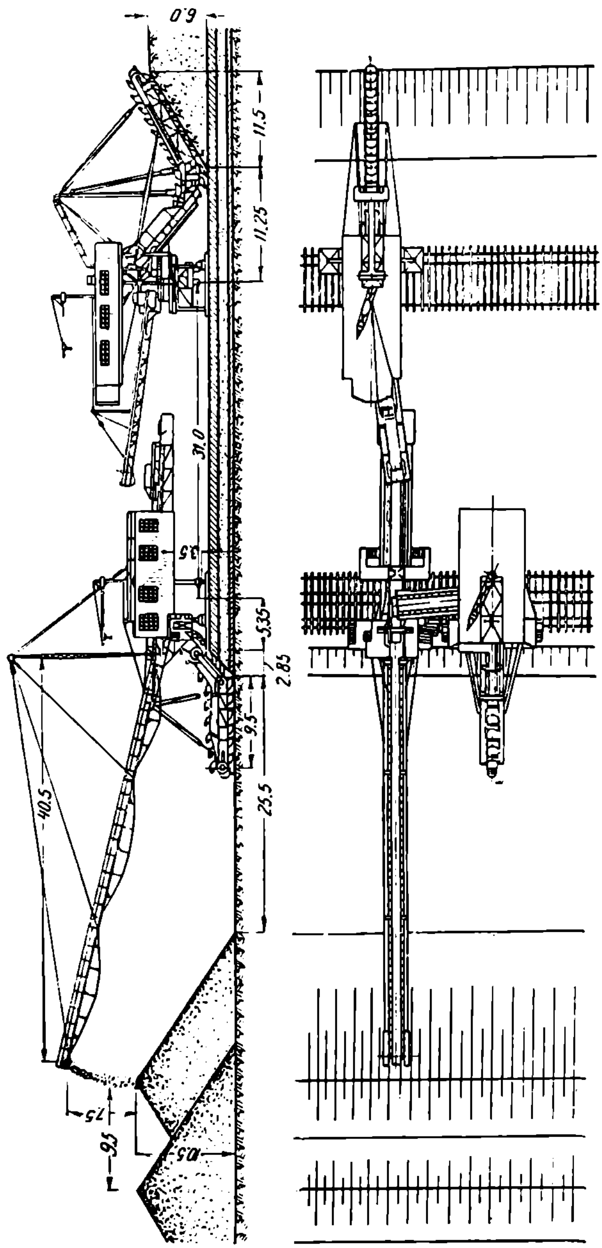



*Fig. 445.* Stacking inside waste dumps at the Lopatinsk phosphorite quarry

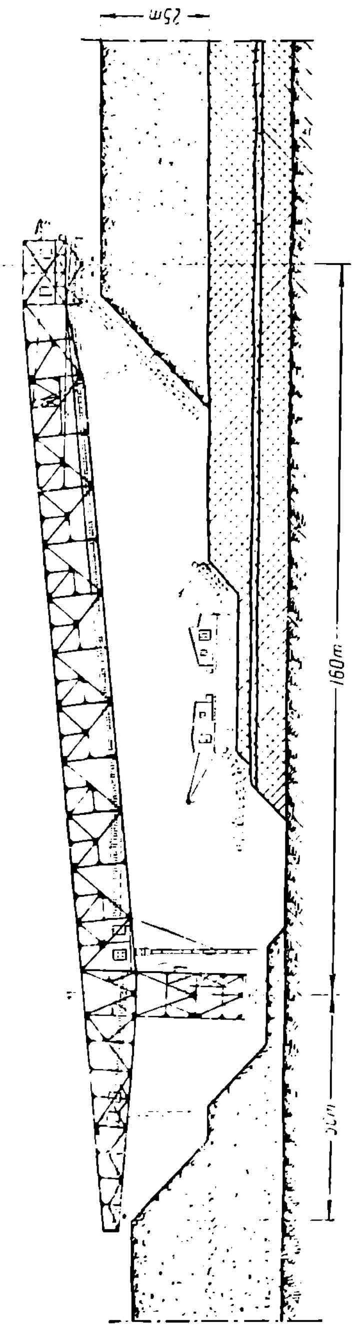



Fig. 446. Overburden bridge

is flat and the bed is very thick.

The first Soviet overburden bridge was commissioned at the Baidakov open-cut coal pit (Ukraine) in 1952.

3. For all their merits, inside waste dumps or spoil banks require definite conditions to warrant their use, and it is by far not in all open pits that they can be built. Much more frequently one has to arrange *outside waste dumps.*

Topography of the land permitting, waste dumps should be sited in lowlands or on valley slopes (see Fig. 407).

Quite often, however, flat land is allocated for waste dumps. In order to obtain the initial specified height of the dump in this case, an inclined fill is built up by a power shovel, or else a timber trestle of required length.

The dump may be made loom- or fan-shaped. In the first instance (Fig. 447), the spoil-loaded and empty trains are not delayed by oncoming traffic, and the dumping front is very extensive. Fig. 447 shows the initial fill (or trestle) from which dumps are built up on both sides, first in fanlike fashion, until it is possible to link the ends of the track lines and form a loop.

The fan-shaped pattern of waste dumps is given in Fig. 448. *Turning point A* is the point from which the tracks radiate. The dump has two independent tracks—*1* and *2*—to provide two discharge sectors.

The loop pattern involves longer train trips than the fan-shaped, but the loaded and empty trains have to meet with no oncoming traffic and the discharge front line is

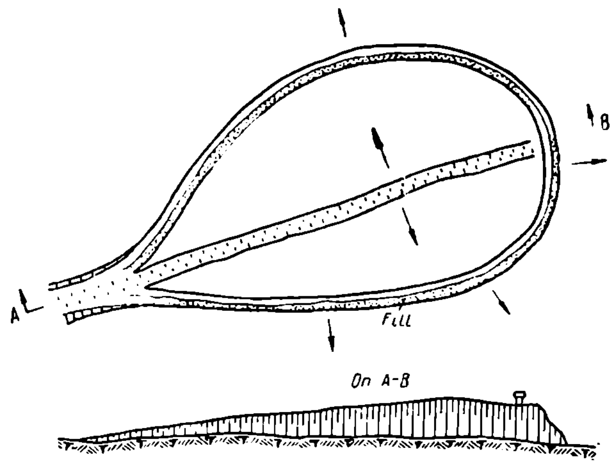



*Fig. 447.* Loop layout of a waste dump

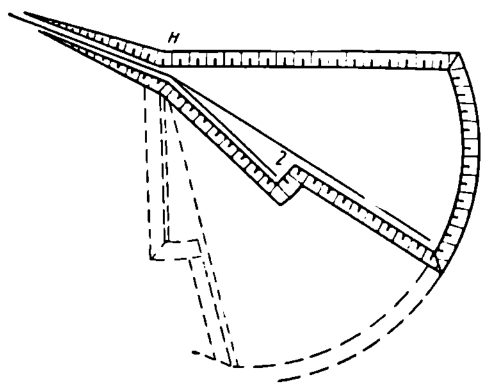



*Fig. 448.* Fan-shaped layout of a waste dump

therefore quite extensive. That is why in big open pits preference is shown to the loop scheme.

The dumps may also be double-benched. That happens when, after a certain area has been covered with the spoil, another dump is started, with the material being discharged in the same order atop the first one. Fig. 449 is illustrative of a very large waste dump at the Korkino brown coal open pits (Urals). To ensure the required storage capacity (and in this case a very high one) the dump is built in two benches, with partly a looplike and partly switchback layout of railway tracks. In Fig. 449 the numbers indicate the points of arrival of spoil-loaded trains.

The principal means used for transporting the overburden to the outside waste dumps of large open pits are trains of large-capacity dump cars drawn by heavy electric or steam locomotives.

However, the use of dump cars in spoil banks entails the following difficulties. As heavy cars are unloaded automatically, railway tracks cannot be laid near the edge of the dump because that would make them unstable. If the tracks are laid away from the edge of the slope, part of the unloaded spoil is dumped on it and thus impedes

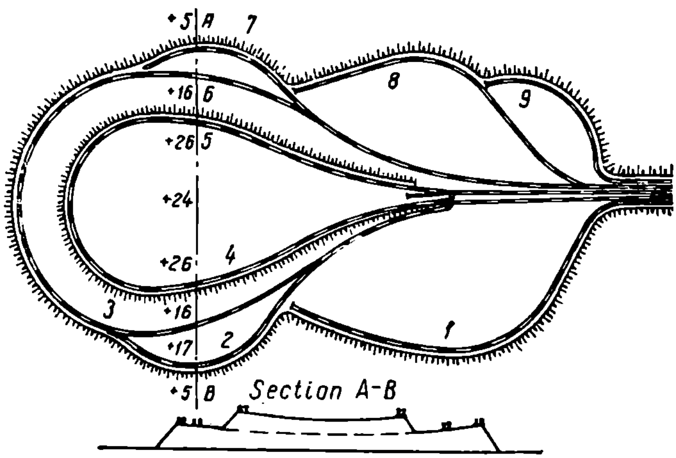



*Fig. 449.* Two-benched waste dump

the discharge of the next cars. That necessitates measures to remove the spoil from the edge of the dump. This is done by spreader ploughs and spreader excavators, or stackers.

*Spreader ploughs* operate in the same manner as the snow-ploughs and cast or drop the waste over the side of the dump with a heavy share or mouldboard. They are either pulled by locomotives (Fig. 450) or driven by an electric motor of their own (Fig. 451). When necessary the mouldboard may be raised or lowered. In the U.S.S.R. spreader ploughs are manufactured by the Magnitogorsk Mining Equipment Works and other plants.

*Stackers* or *power shovels* are used for scooping the spoil discharged from dump cars and casting it away at a certain distance.

Waste dumps may be serviced by power shovels of both ordinary and special design.

Fig. 452 shows that the waste, delivered to point *1* and dropped over the side of the dump, is scooped by dipper *2* of an ordinary shovel and recast to the dump occupying an area of 25 metres in width. Consequently, with this method of dump spreading, the labour-

*Fig. 450.* Trailer spreader plough

consuming job of re-laying tracks has to be done only once every 25 metres. Before the tracks are laid, the uneven surface of the dump is levelled out by a bulldozer.

The scheme of a special spreader excavator or stacker is given in Fig. 453. Its frame *a* has buckets attached to it. The buckets scoop up the spoil brought by train *b* and pass it on to belt conveyer *c*. This same machine is used for levelling out the dump surface prior to track shifting. To do this, the bucket frame is temporarily moved into horizontal position *II*.

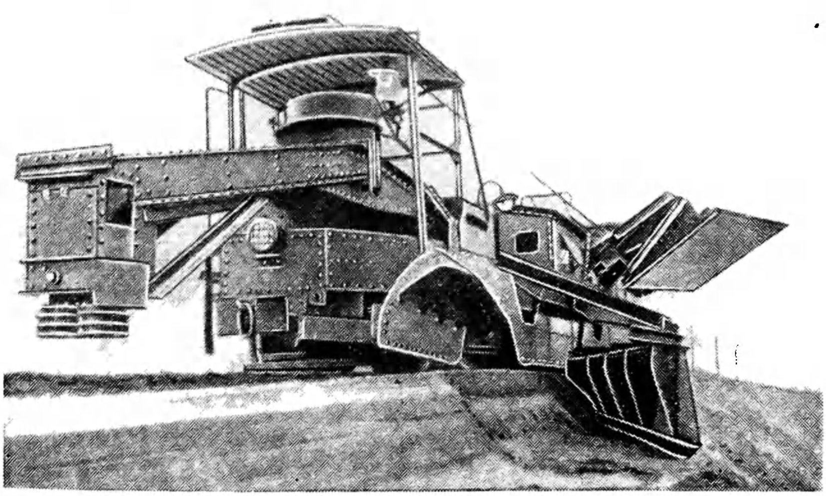


*Fig. 451.* Combination of spreader plough and track shifter

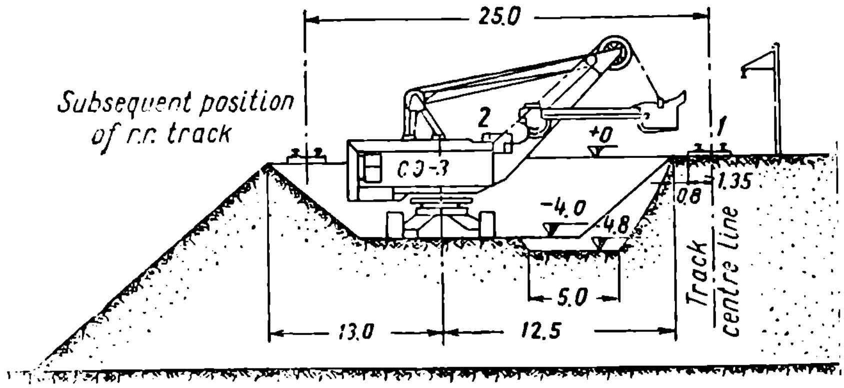



*Fig. 452.* Stacking waste dump by a power shovel

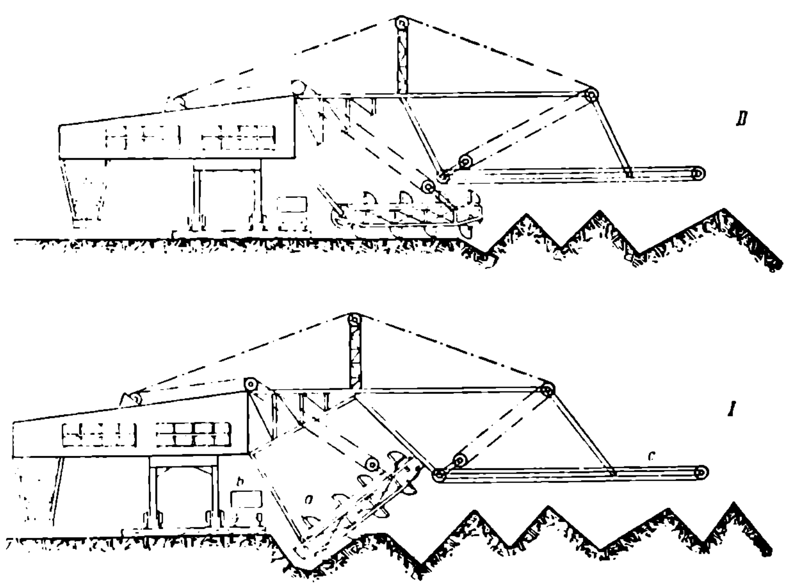



*Fig. 453.* Multi-bucket waste-dump stacker

The use of spreader ploughs and spreader excavators or stackers not only helps mechanise the labour-consuming job of clearing railway tracks and dump edges of the spoil, but also makes it possible to build up higher dumps without the fear of there being any slides, and reduces the number of track-shifting operations to be performed as the dump extends.

Table 24 below lists N. Melnikov's figures for the height and angles of dump slopes as well as labour efficiency depending on the class of the ground and the method of dump stacking and spreading.

*Table 24*

**Waste Dump Characteristics**

| Mode of dump stacking and spreading | Classes of ground | Permissible height of dump, m | Slope angles, degrees | Average output per man per shift, cu m |
|---|---|---|---|---|
| By spreader ploughs | Hard rock<br>Sand<br>Sandy loam<br>Loam and clay | up to 30<br>15-20<br>12-16<br>9-10 | 30-35<br>30-35<br>25-40<br>35-40 | 70-150 |
| Power shovels | Sand<br>Other types of ground | up to 30<br>20-25 | 30-35<br>33-40 | 100-130 |
| Spreader multi-bucket excavators | Sand<br>Sandy loam<br>Loam and clay | 40-50<br>30-40<br>20-30 | 30-40<br>30-35<br>15-30 | 150-300 |

4. Large pits are distinguished by a highly developed railway system, both in the pit itself and in the outside waste dumps, and by the consequent necessity of shifting railway tracks along with the advance of the faces and the extension of the spoil banks. Big cars and heavy locomotives require sturdy rails and long sleepers with large cross-sections.

In such conditions, the transfer and shifting of railway tracks is an extremely labour-consuming operation. For this reason special machines—track-lifting cranes and track shifters—are employed.

A railway crane (Fig. 454) is designed to move sections of tracks, the rails being preliminarily unbolted. In an hour it can move about 100 metres of the tracks to a new place.

By their mode of action, track-shifting machines may be classified into two groups—discontinuous (periodic-action) and continuous types.

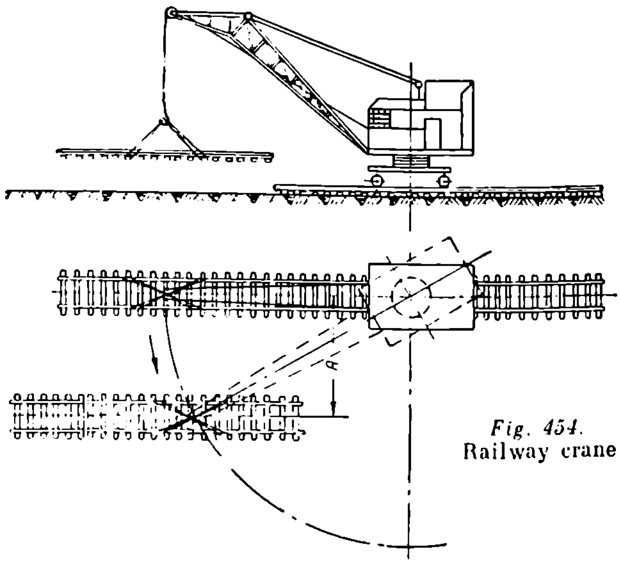


*Fig. 454.*
Railway crane

The first group includes the machine illustrated in Figs 455 and 456, manufactured by the Karpinsk and Magnitogorsk machine-building plants in the Urals. The track shifter has a heavy wheel-mounted platform standing on rails to be shifted. An internal combustion engine of either 32 or 73 hp, designed to drive all its mechanisms, is installed on the platform. Under the platform are special tonglike catches *a* which grip the head of the rail. In addition to this, the platform carries a gear wheel engaged with rack *b*. The bottom end of the rack is hinged to shoe or saddle *c*. The operation starts with the rails being gripped by the catches (*I*). Then, with the rack standing almost vertically, the gear wheel is made to rotate so that the machine, together with the engaged rail track, is raised into position *II*, developing a lifting force of 25 tons. Ultimately, the machine is moved laterally, swinging in an arc around saddle *c*, as a fulcrum (position *III*). A track shifter of this type weighs around 5 tons. Depending on the condition of the tracks and the properties of the ground, the tracks are shifted 0.4-0.8 metre in one operation or "step". Following this, the machine moves 10-20 metres ahead and the shifting operation is repeated.

The main parts of a *continuous* track-shifting machine are the heavy rollers with shaped surface. Drawn apart, they can be lowered onto the rail and then, after being squeezed together, tightly grip the head of the rail on each side of the web.

The spreader plough described above (see Fig. 451) can also serve as a track shifter. The left side of the picture clearly shows the gripping

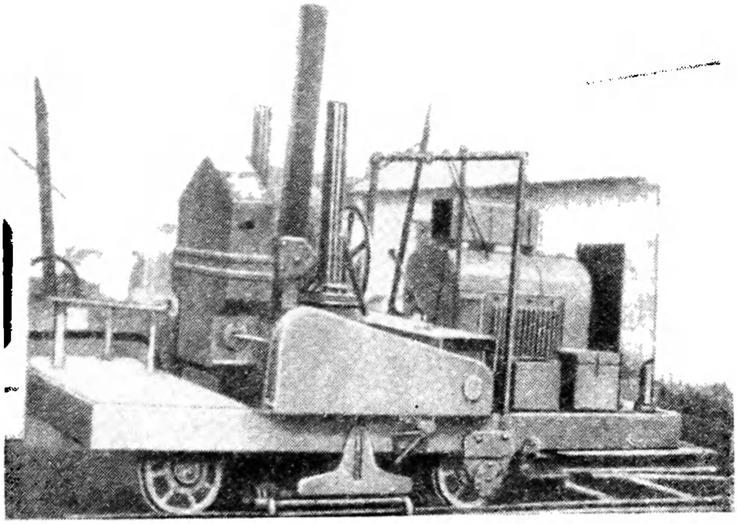


*Fig. 455.* Track shifter

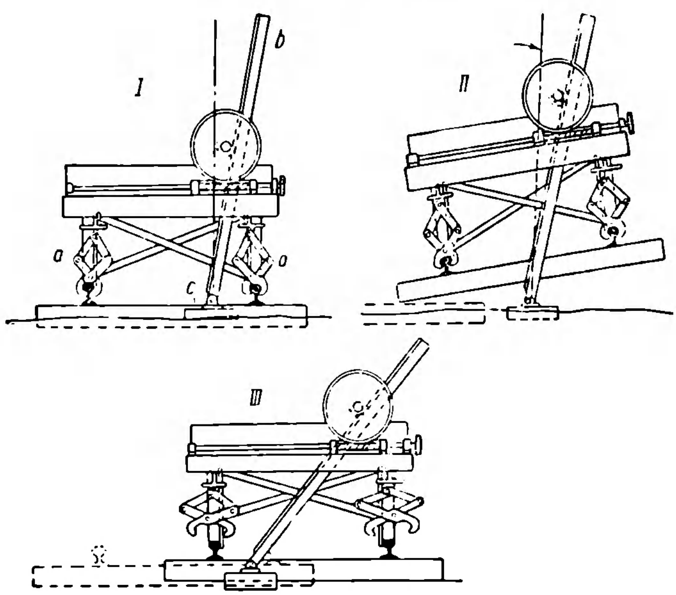



*Fig 456.* Discontinuous (periodic-action) track-shifting machine in operation

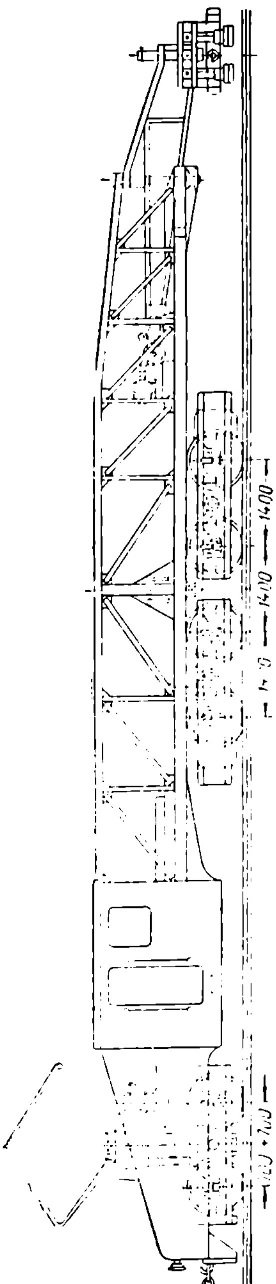


*Fig. 457.* Continuous track-shifting machine

rollers in a raised position. When rails are engaged by the rollers and raised a little, the heavy member they are mounted on is moved laterally with sufficient force not only to bend the rails but also to shift the whole track section together with the sleepers which, in the case of mechanised shifting, are particularly rigidly connected to the rails. The machine then moves forward, with the bending and subsequent shifting of the rails (a few decimetres) going on continuously. A diagram of one of the continuous track-shifter types is given in Fig. 457. Mechanical track-shifting machines are not suitable for the re-laying of railway switches and frogs, this being done by cranes.

The more the surface of the ground is level, the easier it is to shift tracks and the more reliably the whole track-line is laid. What is more, it does not then require additional sleeper tamping. The dump surface is levelled out by a spreader excavator of the design illustrated by Fig. 453.

## 8. General Open-Pit Layout Patterns

The general layout of pits vary widely, this depending on the size and the geological structure of the deposit, land topography, the envisaged scale of open-cut work and the available transport facilities and other equipment. They are classified chiefly in accordance with the distribution of waste dumps and the arrangement and development of the transportation system.

Characteristic from this point of view are the following typical open-pit layouts.

*Pits with Inside Spoil Banks*

1. The pit layout is maximally simple when the overburden is *overcast* directly to inside waste dumps. A series of such layouts are shown in Figs 418, 419 and 443.

Down to a certain depth open pits can be worked with the overcasting or course stacking of the overburden even in steep deposits. Thus, V. Popov gives the following data on open-cut mining in a steeply pitching 20-metre-thick seam in the Kuznetsk coal fields (Fig. 458). After loose silting deposits have been stripped by an ЭШ-4/40 power shovel operated as a dragline, working trench *1* is driven. The overburden is first overcast to temporary waste dump *2* and, later, to permanent spoil bank *3*. All these operations are performed by one and the same shovel over sections extending 100-150 metres. The coal is excavated first in one bench, 18-20 metres high, prepared by the drivage of a working trench, and is loaded into dump-body trucks. The second coal-benching bank is worked by the below-grade digging method, by a shovel installed on the first bank. The shovel dumps coal into piles, from which it is loaded into dump trucks. Daily output may thus be as high as 1,500 tons. The area of working faces needed for the production of this amount can be prepared in 6-8 months.

2. A general idea of a pit layout requiring the building of inside waste dumps with the aid of an *overburden bridge* (see Fig. 446) was given earlier in this chapter.

3. In the mining of extensive open-pit areas and flat or very gently inclined deposits it may happen that the overburden is transported to spoil banks in railway *dump cars*, though the spoil banks themselves lie inside the worked-out area.

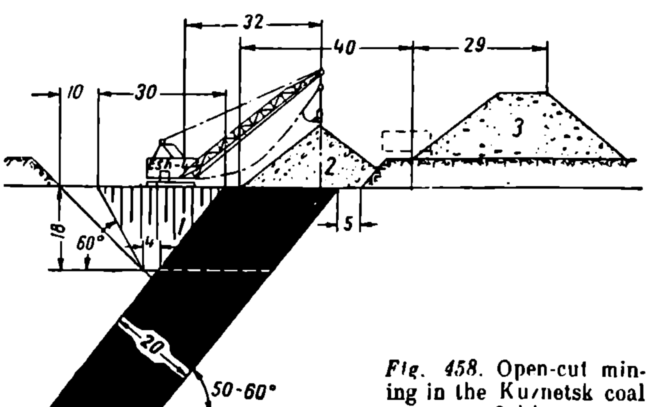



*Fig. 458.* Open-cut mining in the Kuznetsk coal fields

### *Open-Pits with Outside Waste Dumps*

The layout and development of the railway track system, as well as of other means of transportation, may vary considerably. Some typical examples are given below.

4. Figs 459, 460 and 464 are illustrative of a pit layout distinguished by the *parallel* disposition of stripping and production banks and their connection with the outside road system through *trenches*. Fig. 459 depicts stages *1-2-3* of the gradual development of the pit. Stage *3* (full development of the pit) is distinguished by the fact that the spoil excavated in overburden bank $h_1$ is hauled to *outside* waste dumps via stripping trench $RK$, while the mineral mined in bank $h_2$ is transported to the terminal or dressing mill through production trench $R_1K_1$. The front lines of stripping and production faces advancing towards the boundary of the pit always retain their reciprocally parallel attitude. The development of the pit implies the following operations: driving of stripping trench *1*, then stripping operations at large and subsequent cutting of production trench *2*, and after that starting of phase *3* (actual mining of the mineral). These operational stages must be coordinated in time. The location of trenches with respect to the working faces, shown in Fig. 459, is given roughly. Actually, the trenches may be traced differently, though their purpose will remain unchanged. This depends on the topography of the country and the location of waste dump sites and terminals. The outlines of the pit may be irregular and not rectilinear, the decisive factor in this being the geological structure of the deposit.

Fig. 459 depicts overburden bank $h_1$ and production bank $h_2$. If the overburden and mineral deposit are of considerable thickness, there may be not one but two or many such banks. In such cases, special access ramps to individual banks are branched off the trench tracks.

The layout in Fig. 459 is distinguished by the fact that the lead waste and production rear-lines are laid down in *separate* trenches. In a pit worked at a great depth this method, though very convenient because it makes independent operation of waste and production lines possible, would require too great a volume of earthwork in driving trenches. In this instance, therefore, both lines can be laid in one trench.

The outgoing or exit tracks may be laid on the edge of the pit, on berms of sufficient width (Fig. 460). Though this method requires a somewhat smaller amount of earthwork than the preceding one (since there are no trenches outside the pit area), it involves more development work, for the tracks have to be laid on the edge of the deeper part of the pit. Furthermore, the total length of the pit

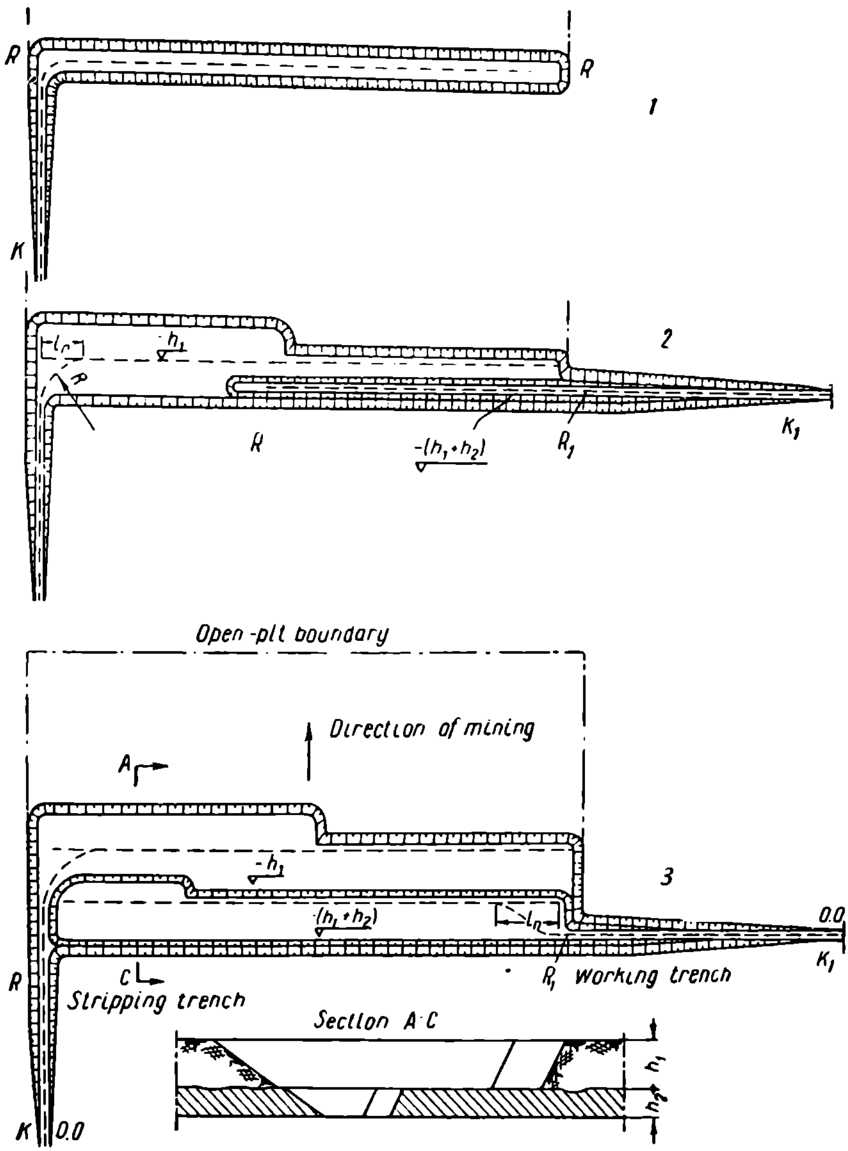



*Fig. 459.* Layout of a pit with separate trenches

must be sufficient to allow the construction of roads with gradients conforming to the type of transportation adopted.

Deeper pits permit the use of the *switchback* system of track connection. Fig. 461 is illustrative of a switchback layout of tracks

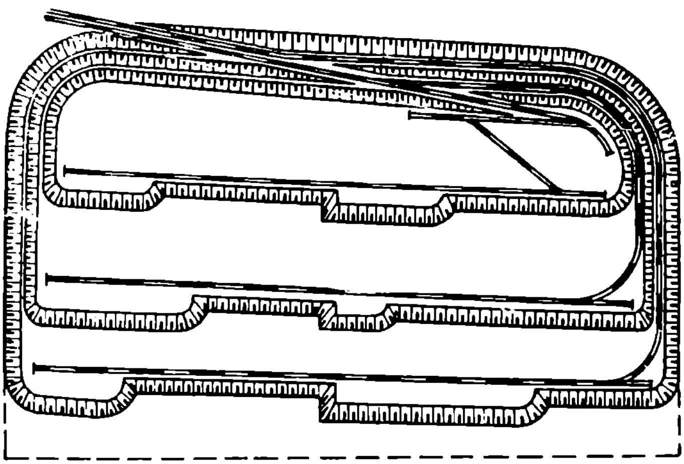


*Fig. 460.* Layout of a pit with outgoing (ramp) roads running along its edges

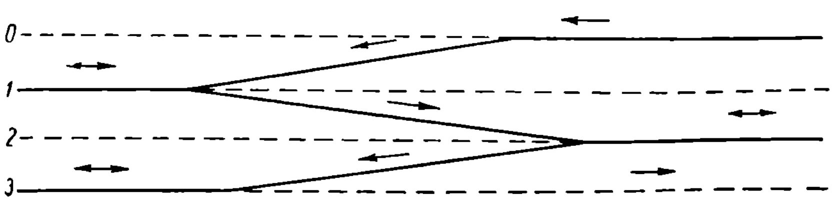



*Fig. 461.* Switchback layout

in a vertical projection. Numbers *0*, *1*, *2*, *3* mark the levels of the floors. Thus, to get from level *0* to working floor *3* a train must follow the route marked by the arrows. Switchbacks entail a big loss of time for shunting operations. Each switchback should be sufficiently long to hold a train and railway switches, or else to allow enough room for motor trucks to turn round.

In deeper pits, roads (both rail and motor) may also be arranged to *spiral* round the periphery of the pit (Fig. 462). The time to start mining the benching bank by a series of parallel strips is when the spiral trench driven from the ground surface round the periphery of the pit reaches the working level. To open a new horizon, the curvilinear trench is deepened still more, and so on. As the result of this, spiral roads encircle the entire portion of the deposit allotted for open-pit work.

There may be instances when, because of mountainous relief, the rail tracks or motor roads connecting the pit with the terminal

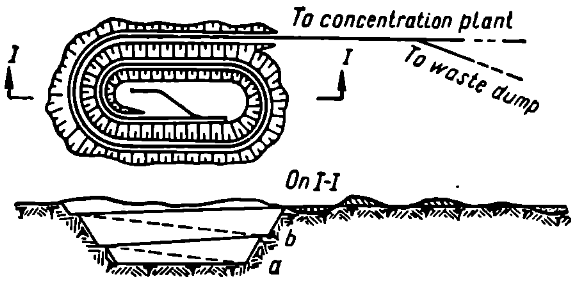



*Fig. 462.* Spiral road layout in a pit

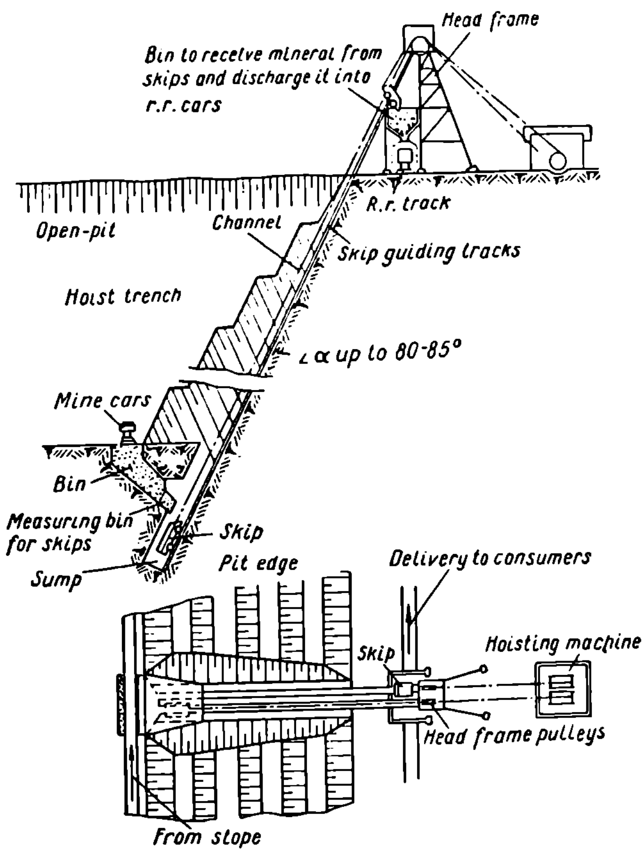



*Fig. 463.* Inclined hoist layout

or dressing mill, run *below* the working faces (for example, Figs 407 and 409). In this event both the spoil and the mineral have to be *cast* or *dropped* over the side to the bench below. And then too, depending on topography, ramp roads—common, switchback or spiral—may be used.

5. The deeper the pit and the smaller the area of the deposit in plan covered by it, the more inconvenient extensive outgoing (exit) trenches, switchbacks and spiral roads become. In such cases recourse is made to *inclined hoists* equipped with cable (Fig. 463) or conveyer plants. To deepen the pit to the height of a fresh bench a dug hole is bored at the bottom, near the hoist plant, and is fitted with a hoist. After that one proceeds with the work of extending the hoisting operation to a new level.

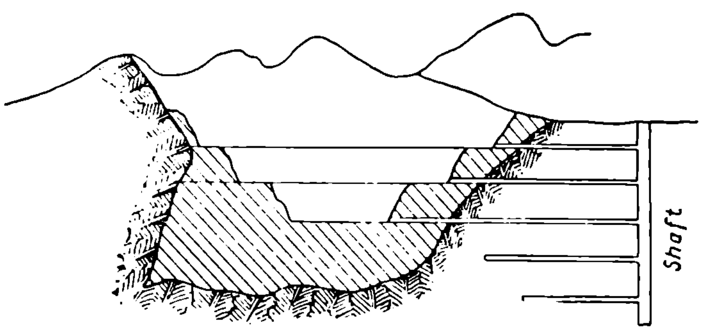



*Fig. 464.* Open-pit and ground-surface connection by underground openings

6. The layouts of open pits serviced by nonwheeled transport facilities have been given above by scrapers in Fig. 419 and by jib cranes for moving cut-stone blocks in Fig. 436. Surface mining of placers will be dealt with in Sections 10 and 11.

7. Sometimes the mountainous relief of the country allows us conveniently to connect the pit, worked by ordinary benches, with the ground surface through underground openings (Fig. 464).

## 9. Driving of Trenches

In open-cut mining it is necessary to make permanent trenches (ingoing or entrance and outgoing or exit) to connect the pit with the ground surface, and also working trenches to develop the banks for mining and other purposes. These trenches may be horizontal or inclined, with the gradient suiting the transportation system adopted.

With electromotive traction, the trench gradient should not exceed 0.030 (in extreme cases 0.040); with steam locomotives it must be not more than 0.020 (maximum 0.025) and for automotive transport not in excess of 0.080. Trenches equipped with belt conveyers must have a gradient of 16-18°. The cross-section of the trenches likewise depends on the transportation system, land topography and the height of banks.

The bottom width of entrance and exit trenches for transportation over standard-gauge railway tracks should be 7.5-8 metres for single tracks and 12 metres for double. The slope angles should not surpass those of repose for the ground in question.

The *driving of trenches* should be mechanised. They are usually made by power shovels and draglines. The power shovel is suitable for the direct excavation of loose ground or for digging hard rocks that have been preliminarily blasted. In both instances the best thing is for the shovel to unload the spoil directly into a dump near the trench (Fig. 465) or into a dump-car running either on the ground surface or on the floor of the upper bank. This is called *up-grade loading*. In certain cases such shovels may be used successfully not only for cutting trenches, but also in pit banks, for this makes it possible to reduce the number of haulage levels. In the latter case it is convenient to use an extended boom with a dipper of somewhat smaller capacity.

Fig. 466 illustrates how a trench is driven by two slices in three cuts. During the first cut *1* the shovel loads the rock into cars *a* spotted on the ground surface; in the second *2* the loading is done into cars standing on the floor of the first cut, and in the third *3* into cars positioned at the bottom of the trench itself. Deep and wide trenches can be dug by a still greater number of cuts.

Draglines are particularly useful and handy in making trenches in loose ground, with course stacking along the side of the trench. Because of their long boom, they ensure greater digging depth, larger dumping radius and discharge height.

Inasmuch as trenches are long but relatively narrow excavations, they are sometimes made by pin-point blasting. With this method (Fig. 467) two rows of explosive charges are fired, one immediately after the other. The rock shot up by the first blast is thrown aside by the second. The method has its merits: simplicity and speed with which work can be done in rocks of any strength; but there are also disadvantages: difficulty of obtaining the desired shape of the opening, additional clean-up work by power shovels, and greater cost of excavation than when it is done by earth-digging machines or hydraulicking. In open-cut work this method is hardly to be recommended.

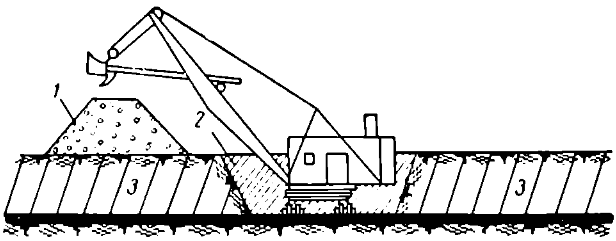



*Fig. 465.* Cutting a trench by a power shovel

*1*—waste dump; *2*—working face; *3*—overburden

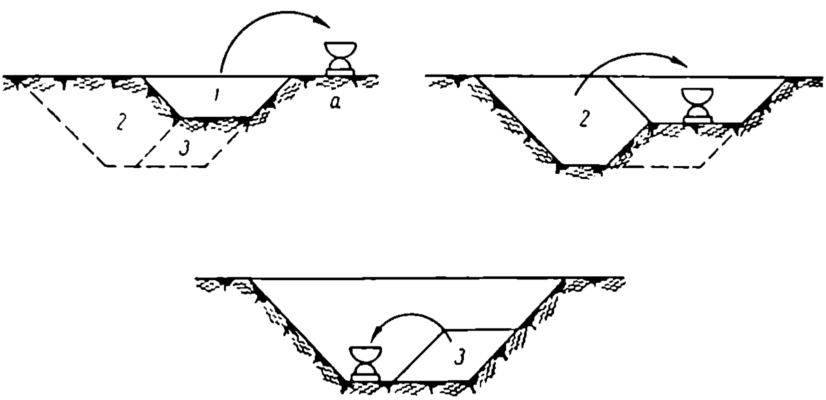



*Fig. 466* Driving a trench in three cuts

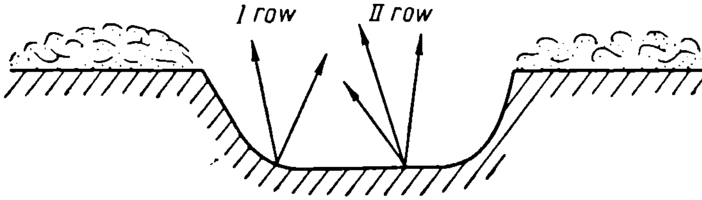



*Fig. 467.* Driving a trench by pin-point blasting

## 10. Hydraulicking in Surface Mining

In principle, earthwork generally implies two basic operations: detachment of the rock from the original solid mass and its subsequent removal to another place. Both these operations can be effected with the aid of water: dislodging of the rock by a powerful jet and displacement by a stream of water. These methods are called hydraulic earthwork. In certain conditions hydraulicking may also be applied in mining open pits.

Loose, friable ground, especially that in which there is no adhesion between individual particles, such as sands, sandy loam, coarse gravel, etc., lends itself most readily to water action. Clayey ground is

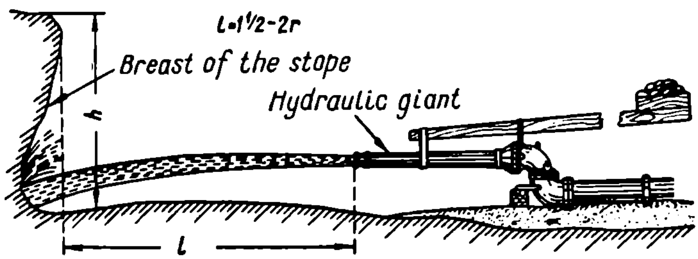



*Fig. 468.* Hydraulic mining of a gold placer

less amenable to washing. Rocks, still more difficult to wash (for example, soft shales), may preliminarily be loosened by blasting or by tractor planers, but this tends appreciably to lower the efficiency of hydraulicking.

Hydraulic mining was introduced in Russia (Urals) in the 1830's in washing of auriferous sands. Under this method, water under pressure is delivered to the working place and its powerful jet, issuing from the nozzle of a hydraulic giant (hydraulic monitor), is pointed at the face to wash the rock (Fig. 468). The barren rock or waste is carried away to the dumps in the valley, while the mined auriferous sand is brought by water to a concentration plant where gold is recovered. Effective hydraulicking requires an adequate supply of water under pressure. Primarily, the hydraulic method was applied in the mountainous regions where a *natural water head* could be obtained. Later on, with the growing use of electric power in mining industry, it became possible to make wide use of an *artificially created* water head, and this has considerably widened the application field of hydraulicking.

Today hydraulicking in surface mining is not only widely practised in working of gold placers but also in extracting other minerals, chiefly in stripping.

It is best when washed ground, mixed with water (pulp), is carried away from the working place by gravity (Fig. 469, *A*). However, if local topography makes that impossible, the pulp has to be brought up (Fig. 469, *B*) by a hydraulic elevator or a pump dredge.

The *hydraulic giant* (monitor) (Fig. 470) is so designed that the water jet issuing from its nozzle may be pointed in various directions and raised and lowered in the vertical plane, this being made possible by a vertical fulcrum (axis of rotation) and a sliding ball-bearing joint. The balance of the giant is ensured by a counterweight. Water consumption by the giant can be regulated by nozzles of different diameters. Soviet plants manufacture hydraulic monitors with inlets measuring from 150 to 350 mm, provided with sets of nozzles from 50 to 125 mm in diameter. The effective range of the jet depends on the head, as follows:

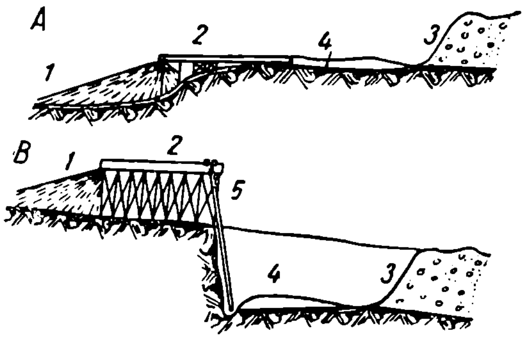



*Fig. 469.* Hydraulic mining layouts
*A*—with pulp carried by gravity; *B*—with pulp lifted by a hydraulic elevator; *1*—waste dump; *2*—sluices; *3*—placer; *4*—inclined ditch at the bottom of the placer; *5*—hydraulic elevator

| | | | | | | | |
|---|---|---|---|---|---|---|---|
| Head, metres.... | 20 | 40 | 60 | 80 | 100 | 120 | 150 |
| Effective range of the jet, metres.... | 10 | 21 | 31 | 41 | 52 | 62 | 78 |

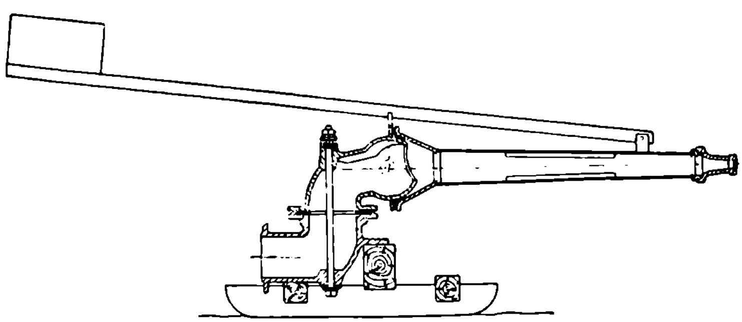


*Fig. 470.* Hydraulic giant

Detailed tables listing relationships between the water outflow velocity and its consumption with various heads and nozzle sizes may be found in a handbook compiled by N. Melnikov.

Citing B. Shkundin, S. Shorokhov gives the following figures. A jet issuing from a 90-mm nozzle at an effective head of 75 metres and water consumption of 800 cu m per hour performs at various distances as indicated in Table 25.

*Table 25*

**Hydraulic Giant Performance and Water Consumption per Unit of Material Washed**

| Distance between nozzle and working place, m | 5 | 10 | 15 | 20 | 25 |
|---|---|---|---|---|---|
| Monitor performance, cu m of ground per hour | 100 | 93 | 74 | 48 | 18 |
| Water consumption per cu m of ground, in cu m | 8 | 8.6 | 10.8 | 16.7 | 44.5 |

As we see, at a given head and with the distance between the nozzle and the working place increasing, the efficiency of the monitor declines quickly, while water consumption per unit of material washed increases sharply. For this reason, the monitor should be set as close to the working face as possible, but the distance must not be inferior to the height of the bank. At the toe of the bank a bottom cut is first made by the monitor's jet and this is followed by the caving and breaking of the ground mass overhanging the cut.

In the hydraulic mining of gold placers, barren cover rocks overlying the productive part of the placer are first carried by the water stream to waste dumps. This is followed by the washing of auriferous sands and other rocks directed to gold-recovering units.

To move fine material, it is enough to have a slope of 0.02, while to move that containing particles of average size the grade should be 0.025-0.05 and more.

The relationship between stream velocity in the *watercourse* and the size of the material entrained is as follows:

| Velocity, m sec | Size of material moved by the stream |
|---|---|
| 0.08 | Wears away fine clay |
| 0.15 | Lifts fine sand |
| 0.23 | Carries coarse sand |
| 0.30 | Moves fine gravel |
| 0.60 | Moves pebbles 25 mm in diameter |
| 0.90 | Moves pebbles of egg size |
| 1.60 | Moves stones 75 to 100 mm in diameter |
| 1.90 | Moves stones 150-200 mm in diameter |
| 3.00 | Moves stones 300-450 mm in diameter |

The *hydraulic elevator* (Fig. 471) is a water-jet pump adjusted for lifting sand mixed with water. It can also entrain pebbles and small stones. The efficiency of the hydraulic elevator is very low (usually 5-20 per cent). Therefore, originally this equipment was used where there was abundant water available at a natural head for lifting sand and pulp, usually to a height of 7-8 metres. With the introduction of electric power in hydraulic mining, the elevators were replaced by dredge pumps.

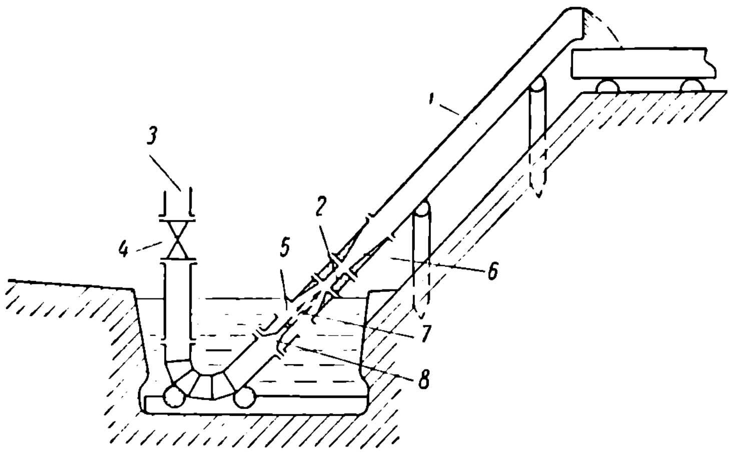



*Fig. 471.* Hydraulic elevator

*1*—pressure (delivery) pipe; *2*—throat; *3*—water under pressure; *4*—slide-valve; *5*—pulp delivery; *6*—dilator; *7*—mixer; *8*—nozzle

The *dredge pump* is a centrifugal pump driven by an electric or diesel motor, especially designed for the suction of water mixed with ground with rock pieces whose diameter should not exceed 0.5 of the diameter of the dredge pump suction pipe.

*Table 26*

**Dredge Pumps for Open-Pit Mining**

| Types | Output by water volume, m³ hr | Head, m | Rpm | Motor rating, kw | Maximum size of rock pieces carried, mm | Weight, kg |
|---|---|---|---|---|---|---|
| 8H3 | 800 | 25 | 730 | 110 | 100 | 2,200 |
| 8H3 | 1,080 | 43 | 960 | 280 | 100 | 2,200 |
| 3ГМ-1 | 1,200 | 43 | 730 | 300 | 180 | 2,775 |
| 3ГМ-2 | 1,400 | 43 | 590 | 300 | 180 | 3,370 |
| 12P-7 | 1,600 | 53 | 590 | 480 | 200 | 4,500 |
| 20P-11 | 3,500 | 42 | 490 | 1,100 | 280 | 9,250 |

Table 26 lists dredge pump models used most widely in open-cut mining. The diameter of the suction pipe is 200-300 mm.

Hydraulicking in earthwork requires very large amounts of water. Adequate water supply and delivery to hydraulic giants are ensured in a variety of ways, depending on whether it is a natural or an artificial head that is utilised. In the first case the water is brought to work places by gravity. At a place selected for its topographic conditions, a reservoir with dams is built and water is conveyed to pressure tanks, sometimes over a long distance, through channels, flumes and troughs.

The *dams* for hydraulicking purposes should be as cheap as possible to build because they are used for a limited period. They have an opening for the outlet pipe, provided with a gate to regulate water delivery and a waterspill for the discharge of surplus water.

*Ditches* are meant to ensure an adequate flow and quantity of water required by the rules of hydraulic engineering. For rough estimates, the following information may be of use.

The ditches are made trapezoid in shape, their bottom width being 1.8-2.2 times greater than the depth. The softer the ground, the less sloping the walls of the ditch should be. The slope ratio in a soft ground is taken at 1 : 1, in stronger ground at 0.5 : 1, and in hard rock at 0.25 : 1. The grade of the ditch is determined by the velocity of the water flow. It must be somewhat slower than that washing away the given ground.

The permissible velocities for different types of ground are:

| Classes of ground | Velocity, m/sec |
|---|---|
| Clay | 0.15 |
| Fine sand | 0.35 |
| Coarse sand, loess | 0.80 |
| Loam and sandy loam | 0.55-0.95 |
| Gravel and small pebble, up to 25 mm in diameter | 1.25 |
| Gravelly ground and cobble | 1.5 |
| Heavy clay | 1.8 |
| Rocky ground | 2.5 |
| Hard rocks | 3.5-4.5 |

The grade of ditches necessary to secure these velocities is computed on the basis of formulas accepted in hydraulic engineering. The useful section and the size of the ditch are determined by the flow and delivery rate of water. There may be water losses in the ditch

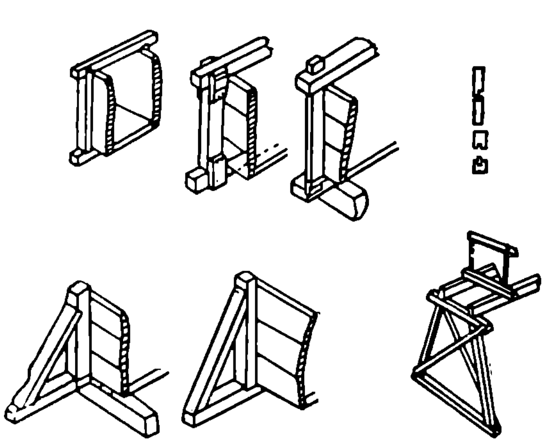


*Fig. 472.* Flumes

through filtration or seepage. If the supply of water in the ditches is limited, they should be appropriately lined.

Over depressions in the ground surface the water is carried via *flumes* (Fig. 472), if the cost of their arrangement and maintenance is below that of the construction of by-pass canals. The service-life of flumes is 8-12 years, but their repairs are expensive (about 0.05 of the initial outlay annually). Steel metal races or troughs may also be used instead of wood flumes.

In places with considerable depressions, where flumes on high trestles would be too expensive, water may be supplied through pipelines by gravity, at a rate of around 1.5 m/sec.

The *pressure* or *surge tank* (Fig. 473) is designed to maintain a uniform head of water delivered to hydraulic giants via a pipe. It has a protective screen to hold back objects that may choke the delivery pipe, and an outlet opening at the bottom for discharging mud. To cut down the length of the delivery pipe, the tank is set up as close to the working place as possible, but not at the expense of the water head.

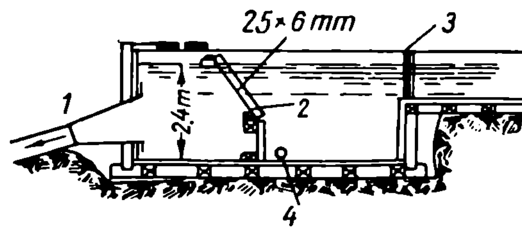



*Fig. 473.* Pressure (surge) tank

*1*—pressure (delivery) pipe; *2*—protective screen; *3*—gate; *4*—outlet opening

*Pumping equipment* is widely used to achieve the desired water head at the hydraulic giant operating in an open pit. The most economic method is the one involving recirculation of water, that is, the use of water partially cleared in the settling ponds at the dumps (Fig. 474). In such cases, it is only the amount of water required to make up for the losses during its recirculation (usually about 10-20 per cent of the total) that has to be drawn from the reservoir. Pumps employed in hydraulicking have an output of 300-1,450 cu m per hour and a head of 60-90 metres, and are driven by electric or diesel motors.

With *hydraulic transport* the rate of flow in *pulp pipelines* should be such as to ensure the movement of ground particles. N. Melnikov cites the following velocity rates required for the transport of the ground through the pipes with the maximum permissible pulp density (Table 27).

For hydraulic transport by gravity the grades should be as indicated in Table 28.

The above listed basic equipment and facilities used in hydraulicking—monitors, pumps, hydraulic elevators, dredge pumps, pipelines, ditches, flumes, etc.—are employed in various combinations, depending on actual field conditions.

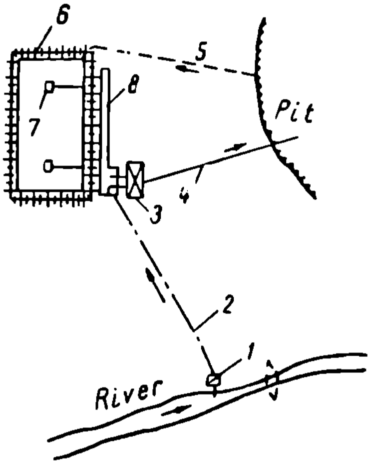


*Fig. 474.* Water supply of a hydraulic plant

*Table 27*

**Rates of Velocity Needed to Transport Pulp Through Pipes**

| Diameter of the pulp pipe, cm | Average velocity in pulp pipeline, m sec | | |
|---|---|---|---|
| | Clay fractions | Sand fractions containing from 70 to 30 per cent of clay | Sand and gravel with low clay fraction content |
| 25 | 1.6 | 2.0 | 2.5 |
| 30 | 1.8 | 2.1 | 2.8 |
| 35 | 2.0 | 2.2 | 3.0 |
| 40 | 2.2 | 2.4 | 3.3 |
| 45 | 2.3 | 2.6 | 3.5 |
| 50 | 2.5 | 3.0 | 3.8 |
| 60 | 2.7 | 3.2 | 4.0 |

Fig. 469 illustrates typical layouts used in the hydraulic mining of an auriferous placer. According to the data furnished by Shorokhov, the results of placer mining are characterised by the following figures. The volumes handled in hydraulic mining vary widely—from 300 to 3,000 cu m per day, depending on local conditions. Small hydraulic plants serviced by pumps and making it easy to shift equipment from place to place may be operated to advantage in the exploitation of small placers only for two or three years. Ditch-supplied hydraulicking, that is, plants fed with water delivered by

*Table 28*

**Grades with Hydraulic Transport by Gravity**

| Type of ground | Grades | |
|---|---|---|
| | for flumes | for earth ditches |
| Clayey ground | 0.015-0.025 | 0.02-0.03 |
| Fine sand (up to 0.5 mm) | 0.025-0.003 | 0.03-0.04 |
| Medium-coarse sand (up to 1 mm) | 0.03-0.035 | 0.04-0.05 |
| Coarse sand (up to 3 mm) | 0.035-0.05 | 0.05-0.06 |
| Gravel and small pebbles (3-4 mm) from 20 to 50 per cent | 0.05 and more | — |

gravity, should be made to work from 5 to 10 years because of the high cost of maintaining water supply and storage facilities. Output per man per shift in small pits is about 10 cu m of ground, in placers of average size—12-25 cu m, and in big ones—40 cu m and even more.

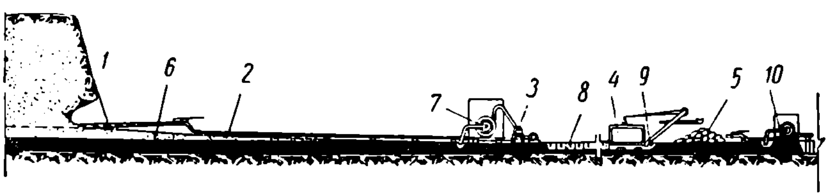



*Fig. 475.* Hydraulic mining layout in an open coal pit

*1*—stripping face; *2*—water pipe; *3*—pulp conduit; *4*—coal-mining power shovel; *5*—coal pile; *6*—material not loosened by the water jet is overcast by the shovel to the monitor; *7*—ЗГМ-1 dredge pump (stripping); *8*—remaining material is removed by a scraper; *9*—coal face; *10*—8113 dredge pump (for coal)

Fig. 475 depicts hydraulic mining in an open coal pit. On the stripping bank ground is loosened by a hydraulic giant and conveyed by gravity to a dredge pump, from which it is carried to a dump by a pulp pipeline. Coal is excavated by a power shovel, but its piles are washed by a monitor and coal itself is carried down to a dredge pump, from which a stream of water transports it through a pipeline to a coal-storage settling site. In the case of hydraulic transport of coal in open troughs, wood flumes should be made to incline 0.05-0.06 and metal ones 0.04.

In the coal-mining industry daily output by a dredge pumping plant reaches 2,000-2,500 cu m and on some days even 6,000 cu m, with labour productivity per man per shift being as high as 30-35 cu m and more (as against 15-18 cu m per shift with stripping by power shovels and transportation of the spoil to waste dumps in dump cars). Coal in the Urals and Siberia is mined with this method 160-200 days in a year, for hydraulicking is seasonal work. In milder climate it can be practised all year round. Electric-power consumption in hydraulicking and hydraulic transport with an artificial water head is fairly high—3-7 kw/m$^3$ and more.

We have said above that it was necessary to cast the spoil discharged by dump cars near the track over the side of the dump. This can be done by hydraulic monitors instead of spreader ploughs and excavators.

Hydraulicking has good prospects in open-cut mining. To make it more efficient, the technique and technology of the method require improvement. What is needed is a readily movable hydraulic giant with controls which would permit discarding the present awkward and difficult hand control, movable dredge-pumping plants, etc. Much also has to be done to organise work better and mechanise ancillary processes.

## 11. Dredging of Placers

Dredges are employed for working placer gold, platinum and tin deposits. The dredge (Fig. 476) is intended for mining ground in water. Therefore, it is a floating digger equipped with a concentration (washing) plant for the recovery of the metal or ore concentrate from the rock. Its digging, recovery and waste-dump stacking equipment is set up on a floating pontoon.

The dredges were originally used for mining deposits under water, with the placer lying beneath the bottom of rivers and lakes. But then dredging proved so advantageous that, in appropriate natural conditions, it pays to erect corresponding hydraulic structures, create artificial water basins over the placer and mine it by dredging. In other words, dredges can be employed both in natural and artificial water basins.

The first dredges were both of single- and multi-bucket types, but the first type has now come into disuse.

The digging mechanism of a dredge works like that of a multi-bucket excavator. The pontoon or hull of the dredge has a long opening for the digging ladder.

The dredge has no self-propelling screw or wheel mechanism (like in a steamer), for, while in operation, it moves very little and very slowly. These movements are effected by tightening the lines anchored either to the bank or to the bottom of the pond and winding on winch drums set up in the dredge (Fig. 477). When in operation, the dredge is held firmly in position by five long cable lines. One of these, the so-called head line, is fixed to the front end or bow of the dredge. While digging, the dredge slowly swings around the point where the head line is fixed, this serving as a pivot, winding in one bow line and paying out the other. Another method of holding the dredge in place during its operation is the use of spuds, long piles pointed on the lower end (Fig. 478), two of which are set at the stern of the dredge. With the aid of the spuds the dredge moves as follows. One of the spuds (for example, $B$ in Fig. 478) serves as a pivot during the digging process, while another ($A$ on the same drawing) serves

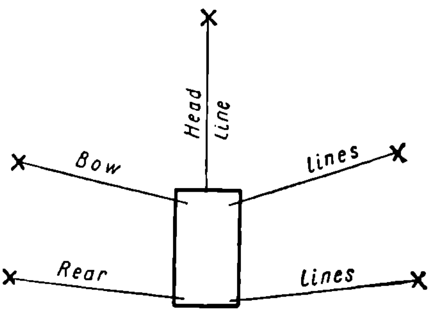



*Fig. 477.* Diagram showing a dredge moving with the aid of lines

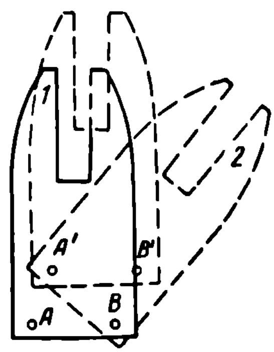



*Fig. 478.* Diagram showing a dredge moving with the aid of spuds

for the forward movement. Two lines suffice to swing or turn the dredge operating with spuds, but large dredges are furnished with several winches and lines to enhance their manoeuvrability.

Fig. 476 gives the outlines of a large dregde manufactured by the Irkutsk Plant. The arrangement of digging, concentrating, stacking and other equipment is explained by the drawing. The waste from the screens of the washing plant is discharged onto the belt conveyer of a long stacker which piles it at a sufficient distance back of the dredge. The waste dumps may be built up on the bank of a pond or in the pond itself, over the worked-out portion of the placer. Thus, Fig. 476 shows that the coarse oversize is transported by the belt conveyer stacker to a high dump, while concentration-plant tailings are discharged from the stern of the dredge at the water level.

The dredges are available in different sizes and capacities. The size of the dredge is usually denoted by the volume of the bucket. The one in Fig. 476 has a bucket of 210 litres.

Table 29 lists the basic characteristics of dredges manufactured by Soviet plants.

There are plans to manufacture dredges of a still larger size (power-driven) with a bucket capacity of 500 litres, designed for very deep digging (down to 50 metres below the water level).

*Table 29*

**Dredges**

| Characteristics | Manufacturing plant | | | | | |
|---|---|---|---|---|---|---|
| | Takhtamy-dinsk | Irkutsk | | Perm | | |
| Bucket capacity, litres . . . . | 50 | 150 | 210 | 380 | 380 | 380 |
| Digging rate (speed of operation in buckets per min) . | 15 | 22 | 24 | 20 | 20 | 22 |
| Maximal depth of digging under water table, metres . . . . | 6 | 9.3 | 11 | 15 | 23 | 30 |
| Length of pontoon, metres . . | 16 | 32.5 | 40.4 | 45.5 | 56.4 | 62 |
| Width of pontoon, metres . . | 8.6 | 15.4 | 18.2 | 22.6 | 22.6 | 24 |
| Pontoon draught, metres . . . | 0.9 | 1.75 | 2 | 2.6 | 2.6 | 2.6 |
| Pontoon material . . . . . . . | wood | steel | | | | |
| Indicated power of motors, kw | 60 | 463 | 850 | | | 1,600 |
| Weight, tons . . . . . . . . . | 109 | 720 | 1,200 | 1,950 | 2,250 | 3,360 |

The great variety of dredge types may be attributed to the fact that they have to be used in mining placers that are different in reserves of the mineral they contain and the depth of occurrence. Dredges for working small placers are equipped with 50-litre buckets, have wooden pontoons and can be manufactured at mine machine shops (except for certain parts). The average daily output of small dredges is in the vicinity of 300 cu m. Small-capacity dredges were introduced by P. Nedoves. These dredges were driven by locomobile and tractor engines.

If electric power is available, electrically driven dredges should be preferred to steam ones.

In dredging, electric power consumption per cu m of sand is about 2-3 kwh.

The sequence of dredging of ground depends on the thickness and structure of the placer, but the usual practice is to work by several horizontal cuts or slices *1*, *2*, *3* (Fig. 479), and much more seldom by

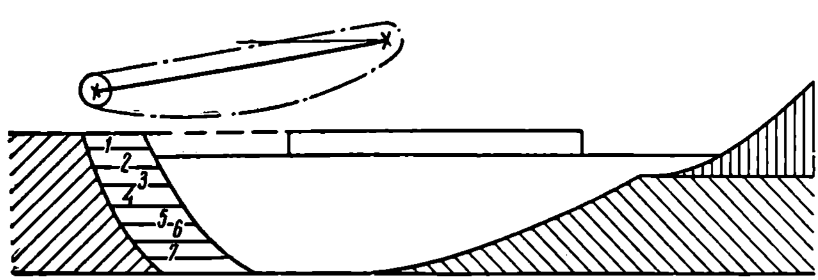



*Fig. 479.* Dredging of a placer by horizontal slices (cuts)

one cut, over the entire thickness of the placer. As required by the depth, the digging ladder is lowered or raised by steel cables which pass over sheaves on the front or bow gantry. Extraction by horizontal slices makes it possible selectively to mine the barren rock and the productive part of the placer. The valuable components of the placer are usually concentrated at the very bottom, on the bedrock. Placers whose bedrock consists of hard fissured ground are not very suitable for dredging, for much metal may be lost in crevices and depressions. Large dredges with heavy buckets are better suited for a thorough "clean-up" of the placer floor than the small ones.

In regions with rigorous climate one serious obstacle to dredging is *permafrost*, which necessitates preliminary *thawing* of the ground. This is done first by removing the vegetative cover and the upper soil layer over the area destined for working by the hydraulicking method or with the aid of earth-digging machines, after which the permafrost laid bare is exposed in summer to solar radiation. To speed up the thawing process, steam, hot, or even unheated water are passed through numerous steel points nailed into the frozen ground.

Dredging is feasible in the following conditions:

1. The placer must have sufficient reserves of "sand" with the average value of metal per unit of volume warranting its working for not less than 10 years and, in the instance of 50-litre bucket dredges, for at least 5 years.

2. The valley slope should not be too steep, not more than 0.02-0.03, to make it easier and cheaper to build artificial ponds.

3. The amount of water available in the working pond must be sufficient for the dredge to operate in.

4. The placer floor must not be too hard, uneven or fissured.

5. The placer should not contain many large boulders.

6. Frozen placers can be dredged only after they have first been thawed.

The advantages of working with dredges are:

1. Immense output by large dredges—up to 12,000 cu m a day and 4,000,000 cu m a year. Small-size dredges handle annually up to 40,000 cu m of ground.

2. High efficiency of labour—up to 50 cu m per man per shift with 210-litre bucket dredges and up to 90 cu m with those of 380 litres.

3. Dredges can be employed in mining poor placers, which it is unprofitable to work by any other method. Large dredges can successfully and economically exploit gold placers with a metal content of but few hundredths of a gramme per cubic metre of ground.

There are numerous cases on record of dredging, which is cheap, being employed in reworking abandoned old dumps of placers, formerly mined by hand labour. In placers previously worked by the underground method, the presence of abandoned mine timber may hamper dredging operations.

In regions with relatively warm winter dredging can be practised all the year round. In rigorous climate the number of working days suitable for dredging drops to 200-250 a year. In these conditions the winter months are usually devoted to major repairs of equipment.

## 12. Dewatering Open Pits

Water *inflow* in open pits may be due to precipitation directly over the area of the pits, or it may come from neighbouring catchment grounds or, finally, underground sources.

Annual precipitation directly over the open pits is rather small, about 400-500 mm in average climatic conditions. This means that the amount of water caused by rain and snow in an area of one square kilometre comes annually to about 400,000-550,000 cu m. But since part of it evaporates and another penetrates into the ground, only about 200,000 cu m remain to be drained. On the other hand, a pumping plant of a very moderate capacity, say, 100 cu m per hour, can dispose of $100\times24\times365=876{,}000$ cu m of water by working continuously throughout the year. Therefore, it is not the total amount of water finding its way directly from the atmosphere that is important for estimating the rated capacity of water-disposing equipment in open pits, but rather its concentrated influx during heavy rainfalls and the period of thaw.

A big danger is presented by water coming to the pit from adjacent catchment areas. The danger is prevented by diverting the streams and building dikes and ditches.

Underground water penetrates into the pit from its edges, where aquifers become exposed, or, in the case of pressure water, through the bottom of the pit. Sometimes ground water is so abundant that the deposit has to be *dewatered first* to ensure proper operation of the pit.

*Discharge* facilities are provided to dispose of water in open pits. The best thing possible is when water can be made to run by gravity down to neighbouring valleys and ravines. For this purpose, local topographic conditions permitting, underground drain tunnels (adits) are driven. If this proves impossible, the inflowing water is diverted by ditches to the lowest point, where a header is arranged with a pumping plant. In large open pits there may be several such plants. Because of the constant extension of development and production workings, the floor of the pit is usually uneven and gradually becomes deeper and this very often complicates the delivery of water to pumps through the ditches cut in the floor of the pit. In such cases a vertical shaft is sunk in the vicinity of the pit, in the lowest portion

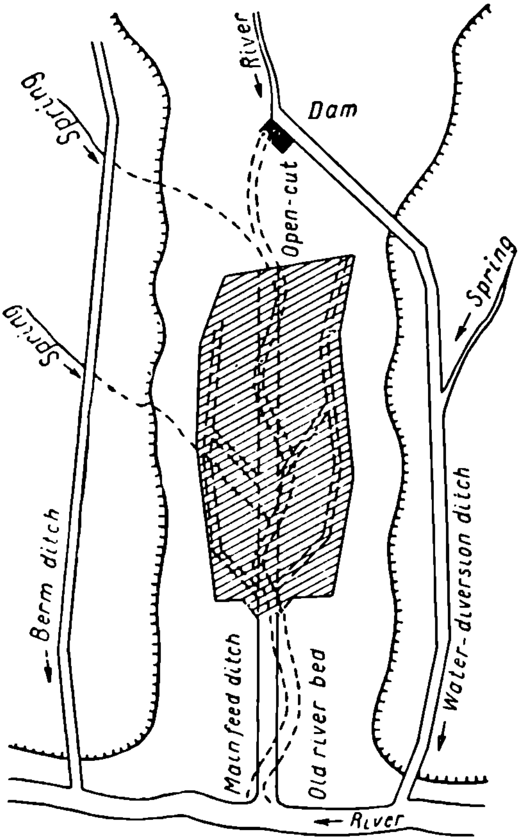


*Fig. 480.* Diversion of water in surface mining of a placer

of the deposit, and from it a series of openings, sloping slightly towards the shaft, are driven at a level somewhat below the envisaged depth of the pit. The job of these openings is to collect water and thus drain the pit. Usually in these cases it suffices for water to seep through the natural crevices in the rock. If necessary, however, water can be made to run off through boreholes or the inclined or vertical openings driven from the bottom of the pit. A sump and a pump room are built near the shaft to bring water to the ground surface.

Some examples of diverting *surface* water from the pit area are furnished by the open-cut mining of auriferous placers. The methods employed are analogous to those used in the drainage of placers worked by underground methods, described in Chapter XX. A typical pattern of water disposal from a placer area is shown in Fig. 480. The section of the mined placer (hatched in the drawing) is located in the valley of a river flowing into a bigger one. To divert the small river a *drain ditch* is cut. At the site of the diversion the former river-bed is *dammed.* The drain ditch is also used to intercept water originating from springs and temporary streams flowing down the slopes of the valley. When the valley slopes are not even, the ditch—which should be located as low as possible—is run along the gentler slope. On the other, steeper slope, a *hillside* or *berm ditch* is arranged to intercept and divert spring and meteoric water from the adjacent grounds.

A permanent ditch, whose depth is to exceed that planned for the pit, is made along the former river bed. To facilitate the dewatering of the working section, provisional ditches (shown by dash lines) are dug when necessary. The cross-section of the outfall ditch is determined by the actual water flow in the river, while hillside ditches

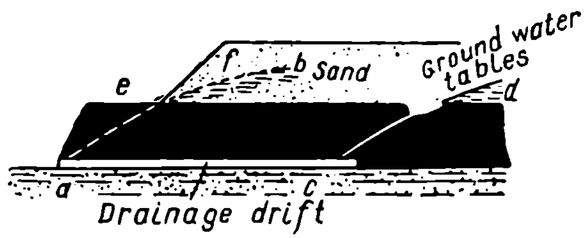



*Fig. 481.* Dewatering of a coal seam through drainage workings

are 0.7-1 metre deep and 0.7-1.8 metres wide at the bottom and 1-2.5 metres at the top. The permanent ditch should be deep but narrow and have heavy timbering reinforced by braces. When the driving of ditches involves large amounts of earthwork, *trench diggers* should be used.

Measures for *underground* water control are explained below.

Fig. 481 illustrates the working of a strongly aquiferous lignite bed, covered by sand strata and containing abundant water. Depression surface *ab* is formed near the edge of the pit. Wedge-like sector of drained coal *e* lies over the strata of water-saturated coal, and this creates the danger of slides. An analogous situation is seen in the country rock face (wedge *f*). To dewater the working faces, drain openings *ac* may be run. They will collect water within area *cd.* Such openings are made at intervals of a few scores of metres.

In the event the ground is very soft and it is difficult to drive drain openings, *absorbing* or *suction filters* can be made use of. They are in the form of boreholes fitted with filter screens and equipped with pumps. Fig. 482 gives an idea of draining a thick layer of aquiferous sand capping a brown coal bed. It takes four benches with several rows of suction filters to lower the water level by 16 metres. After the dewatering of the first bench, the ground is stripped by an excavator to a depth of 4 metres and the second row of filters is set up. The pipes connecting them are moved together with the centrifugal pumps. Fig. 482 illustrates the drainage of the fourth bench. The stripping job is done by a chain-and-bucket excavator. In the conditions shown in Fig. 482 the intervals between the filters on the first bench come to 120 metres, on the second—80 metres, on the third—

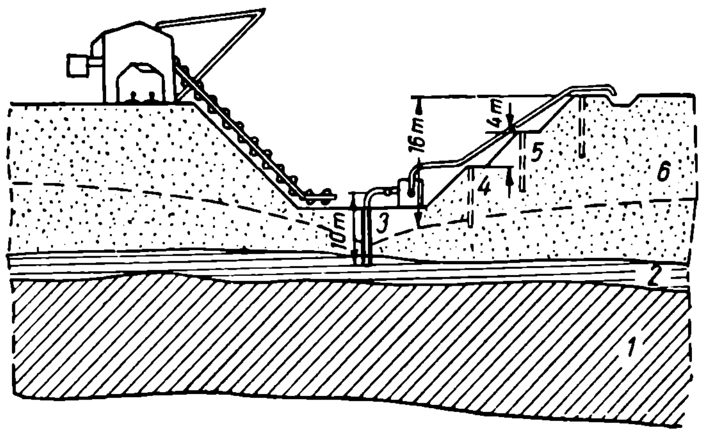



*Fig. 482.* Drainage of an open pit through suction filters
*1*—coal; *2*—clay; *3, 4, 5*—banks; *6*—sand

50 metres and on the fourth—25 metres. The bed is exposed over a distance of 2 km. The water inflow per one filter is as much as several cubic metres per minute.

*Deep-well pumps* have latterly been used for the preliminary drainage of the deposits which are to be mined by the open-cut method and which are badly flooded by underground waters. These are electrically driven centrifugal pumps of a special design, suitable for lowering into boreholes. They have filter screens that let water through but hold back sand particles.

## 13. Safety Measures in Open-Cut Mining

Many of the factors causing accidents in underground work, such as caving of the roof, evolution of the firedamp, accumulation of explosive dust, rope hoists, etc., are absent in open pits.

But injuries may be caused by pieces of rock falling from bench banks, accidents during the transportation and operation of machines, electric shocks, blasts, etc. Therefore, appropriate safety measures in open-cut pits are just as imperative as in underground mines.

To prevent injuries by falling rock pieces, the height of banks and the angle of slopes should be kept within normal limits. The slopes of banks, their edges and berms must be systematically and carefully checked for crevices, jointings and detached slabs, which may prove to be a source of danger to men working in the pit. Such slabs, as well as the snow masses and ice lumps overhanging in winter, must be removed as soon as they are sighted.

When work on the banks is completed, the width of the berm should equal 1/10 of the bank's height and not less than one metre.

To prevent people from falling in, the pits and quarries near settlements and roads should be fenced off.

Drilling and blasting operations should be carried out in accordance with the existing rules and regulations. Special attention should be paid in handling explosives, particularly in winter.

Accident statistics reveal that transport operations are the main source of industrial injuries. Hence the need of strictly observing the safety regulations concerning trains in pits and waste dumps.

It has been proved statistically that conveyers are safer than the wheeled vehicles.

Machinery and transport equipment employed in open pits are as a rule extremely heavy and bulky and, for this reason, one must strictly observe safety rules in setting up and repairing them.

Since electric power is widely used in open-pit work, particular attention should be paid to safety measures against injuries caused by electric current.

The *lighting* of open pits at night is not only necessary for efficient operations but also for safety. Working places and machines should be provided with ordinary powerful electric lamps or special stationary or portable floodlights. Portable lighting mains should be used at the sites of blasting operations.

# SUBJECT INDEX

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Л. Д. ШЕВЯКОВ

**РАЗРАБОТКА МЕСТОРОЖДЕНИЙ ПОЛЕЗНЫХ ИСКОПАЕМЫХ**